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AMERICAN INSTITUTE OF MINING
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(INCORPORATED)

VOL. 76

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HELD IN NEW YORK, FEBRUARY, 1928; PHILADELPHIA, APRIL,
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AMERICAN INSTITUTE OF MINING AND METALLURGICAL ENGINEERS
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PREFACE

This volume constitutes the seventy-sixth issue in the series of TRANSACTIONS of the Institute, of which the first was printed in 1873. In general form and arrangement it follows the plan of its predecessors but in content it is less inclusive, marking in that particular a further step toward diversification of publications to correspond with specialization of members. In addition to the annual volume of TRANSACTIONS and the YEARBOOK which contains the list of members, committee assignments, constitution and similar data, the Institute now furnishes annually three volumes each reporting on the activities of one of its professional divisions, the third in this series having been established within the year. For the year 1928 these volumes were as below.

PETROLEUM DEVELOPMENT AND TECHNOLOGY IN 1927. 844 pp., including the papers read before the Petroleum Division at the Fort Worth, Texas, meeting of Oct. 19-20, 1927, and at New York, Feb. 20-23, 1928. This volume was issued in June, 1928.

PROCEEDINGS OF THE INSTITUTE OF METALS DIVISION. 836 pp., including papers read before the Division at Detroit, Sept. 19-24, 1927, and New York, Feb. 20-23, 1928. It was issued in August.

IRON AND STEEL TECHNOLOGY IN 1928. 353 pp., being the record of the year's work of the Iron and Steel Division and including papers read at New York, Feb. 20-23, and a paper presented before the Institute of Metals Division and the American Society for Steel Treating in joint session at Philadelphia Oct. 10, 1928. Issued in December.

Aside from the Division volumes listed above, the Institute now issues a series of *Technical Publications* in the form of unbound pamphlets arranged in eleven series according to subject. A complete list of those issued in 1928 will be found on page 689 of this volume. From time to time the papers in any one series or on one subject are grouped and published as a special volume under the auspices of one of the technical committees. One such volume, on flotation, was issued in 1928, and a second, on geophysical prospecting, will be issued in January, 1929. The first-named volume was:

FLOTATION PRACTICE. 257 pp., issued under the auspices of the Milling Committee and including papers presented and discussions at meetings held at Salt Lake City, August, 1927, and New York, February, 1928. Issued in November.

In view of the wide distribution of the material in these special volumes to those immediately concerned, and the availability of most of

the separate papers in the series of *Technical Publications*, the Papers and Publications Committee has thought it unnecessary except in a very few cases, to reprint papers from these volumes in the *TRANSACTIONS*. With this opinion the Board of Directors has concurred and accordingly the technical papers and discussions in this volume are almost entirely in fields not covered by the special volumes indicated. In order, however, to facilitate ready finding of any paper published by the Institute within the year, a comprehensive index has been prepared and is printed at the back of this volume. In it will be found reference to every paper printed within the year in any form except a few of which a limited edition was preprinted for use at some meeting and which are being held for later formal publication with discussion. Attention is especially called to the fact that articles and abstracts in *MINING AND METALLURGY* are included in this index. Since all papers issued by the Institute are abstracted in *MINING AND METALLURGY* and the latter is sent to all members, this will make it possible easily to find out the character and content of any paper.

In view of the fact that full accounts of all meetings are printed in *MINING AND METALLURGY* it is not considered necessary to review in this volume the meetings themselves, and the reports already printed are cited instead. For the same reason obituaries are not reprinted here, as has sometimes been done in the past. These various changes in plan of publication have made it possible, without increasing the total expense, to publish and distribute to members of the Institute a much larger volume of papers than at any time in the past, to bring publication fully current, and to furnish in this volume a comprehensive index of all the publications of 1928 within the first month of the succeeding year.

H. FOSTER BAIN.
Secretary.

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For the year ending February, 1929

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¹ Until Feb., 1929.

² Until Feb., 1930.

³ Until Feb., 1931.

Institute Meetings in 1928

The Annual Meeting was held at Institute headquarters in New York City, February 20-23, 1928; for program of meeting see MINING AND METALLURGY for February, page 54, and for complete story MINING AND METALLURGY for March, page 104.

REGIONAL AND DIVISION MEETINGS

Meeting	Place	Date	Report in MINING AND METALLURGY, Page
Joint meeting American Institute of Mining and Metallurgical Engineers, American Society of Mechanical Engineers and National Coal Association	Columbus, Ohio	April 21	May, 241
Joint meeting American Institute of Mining and Metallurgical Engineers, American Society of Mechanical Engineers and National Coal Association.	Philadelphia	April 24	May, 245
Regional meeting.....	Boston	Aug. 29-31	Sept., 390* Oct., 429
Joint meeting Southern California Section, A. I. M. E.; American Mining Congress and California Development Association.	Los Angeles	Sept. 10-13	Sept., 403* Oct., 445
Joint meeting Iron and Steel Division, A. I. M. E., and Lake Superior Mining Institute.	Crystal Falls and Iron Mountain, Mich.	Sept. 7-8	Sept., 413* Oct., 443
Joint meeting Institute of Metals Division, A. I. M. E.; American Society for Steel Treating, and American Welding Society.	Philadelphia	Oct. 8-12	Oct., 448* Nov., 500
Petroleum Division, A. I. M. E.....	Tulsa, Okla.	Oct. 18-19	Oct., 453* Nov., 496

* Preliminary story of meeting containing program.

Necrology

The following is a list of members who died in 1927. It is compiled from reports to the Secretary's office. Biographical sketches published in MINING AND METALLURGY are indicated in the last two columns.

YEAR OF ELECTION	NAME	DATE OF DEATH	ISSUE CONTAINING BIOGRAPHY 1927	PAGE
1902	ADAMS, THOMAS J.....	Feb. 20		
1920	AID, HARRY.....	Jan. 4	March	154
1915	AUMAN, WILLIAM.....	Dec. 6	February*	90
1885	BARTLETT, JAMES HERBERT.....	Dec. 17	June*	285
1908	BEALE, HORACE A., JR.....	Sept. 6	October	446
1914	BELL, ARTHUR F. L.....	July 10	August	364
1920	BOTSFORD, CHARLES WARD.....	Apr. 10		
1902	BRANDES, JUAN FELIX.....	Oct. 29	December	525
1920	BROWN, CHARLES M.....	Mar. 8	May	238
1914	BROWNING, EDWARD.....	Oct. 22		
1883	BRUNTON, D. W.....	Dec. 20	January*	37
1904	BUNTING, DOUGLAS.....	Dec. 15	January*	38
1899	BUSH, B. F.....	July	March*	157
1921	CAMPBELL, ALEXANDER G.....			
1915	CARROLL, FRED.....	May	July	326
1920	CLUNE, DANIEL EDGAR.....	Jan. 1		
1895	CRAFTS, WALTER N.....	June 8	August	363
1924	DARBY, E. H.....	June 11	August	364
1888	DE LA GOUPIILLÈRE, H.....	Jan. 7	April	193
1900	DE LASHMUTT, IVAN.....	Oct. 29	December	526
1914	DE SALLIER DU PIN, RENE.....	Dec. 13	January*	38
1925	DICKERSON, A. V.....	Mar. 26	May	238
1890	DUMBLE, E. T.....	Jan. 26	March	154
1893	EDWARDES, WILLIAM S.....	Oct. 21		
1924	ELDER, PAUL G.....	July 8	September	403
1914	FEUCHERE, LEON.....	Oct. 28	January*	38
1913	FEUST, A. E.....	Feb. 24	April	193
1882	FEUSTMAN, LEON P.....	Apr. 7	June	281
1879	FORD, EDWARD L.....	July 4	August	363
1875	FORSYTH, ROBERT.....	Sept. 11	October	445
1904	GARY, E. H.....	Aug. 15	September	404
1923	GODDARD, HARRY W.....	Sept. 8	October	445
1888	GOODRICH, ROBERT RHEA.....	Aug. 10	October	445
1892	GRASSELLI, C. A.....	July 28	September	403
1914	HAMILTON, JOHN P.....	Jan. 24	April	193
1914	HARDISON, S. J.....	June 29	February*	90
1925	HOOVER, C. C.....	Feb. 7		
1899	HUDSON, A. W.....	July 14	February*	90
1885	HUMPHREYS, A. C.....	Aug. 14	September	403
1895	KANN, WILLIAM L.....	Dec. 22	February*	88
1922	KEFFER, ROBERT.....	Nov. 19	January*	38
1917	KENNARD, E. H.....	November	August*	376
1880	KING, CHARLES F.....	May 25	July	326

* 1928 issue.

YEAR OF ELECTION	NAME	DATE OF DEATH	ISSUE CONTAINING BIOGRAPHY 1927	PAGE
1922	KNUPPE, L. M.....	Jan. 16		
1920	LLOYD, R. MCA.....	Dec. 14	January*	38
1887	MACKAY, WALTER.....	Mar. 19		
1923	MCCORMICK, ROSCOE T.....	Dec. 7	February*	90
1905	MCGEE, JOHN.....	June 12	December	525
1907	McKIM, JOHN W.....	Dec. 31	March*	159
1924	MINTER, LEWIS S.....	Jan. 26		
1927	MITCHELL, ELLIOTT M.....	July 9	May*	240
1914	MITCHELL, JOHN R.....	Nov. 7	June*	284
1882	MOSS, MILTON.....	Dec. 1	March*	158
1906	MURDOCH, JOHN A. W.....	Aug. 21	March*	157
1893	NICHOLS, RALPH.....	Oct. 23	December	525
1903	NORWOOD, CHARLES J.....	Jan. 20	March	154
1920	PARMELEE, CHARLES L.....	Apr. 4	May	238
1900	PATTBERG, OTTO F.....	Oct. 30		
1874	PEARCE, RICHARD.....	May 18	February	89
1906	PETERSON, FRANK.....	Jan. 21		
1874	PORTER, JOHN A.....	Feb. 1	April	194
1891	PULLEN, JOSEPH T.....	Jan. 27		
1920	QUINCY, CHARLES F.....	Oct. 1	December	526
1897	RAND, CHARLES F.....	June 21	August	362
1925	RANDOLPH, A. M.....	December	July*	333
1914	RICE, LUTHER VINTON.....	Nov. 26	January*	39
1920	RIDLEY, FREDERICK WILLIAM.....	May		
1889	RISSMANN, OTTO.....	May 1		
1897	ROGERS, CHARLES EDWARD.....	Nov. 1		
1874	ROLKER, CHARLES M.....	Oct. 14	November	485
1882	ROSE, WILLIAM W.....	December		
1926	ROWAN, R. F.....	Sept. 29	June*	285
1901	RYDER, THOMAS J.....	January		
1918	SCHLOSS, JOSEPH A.....	June 9	August	363
1921	SEAMON, W. H.....	Aug. 2	September	403
1897	SMITH, ALBERT W.....	Mar. 4	November	485
1913	SMITH, CHARLES S.....	Aug. 28	November	485
1909	SMITH, EUGENE A.....	Sept. 7	October	445
1896	STEWART, ALEXANDER.....	May 15	August	363
1914	TANNER, WALLACE N.....		March*	157
1898	TAYLOR, JAMES.....	Dec. 14		
1925	TEALE, F. LAYTON.....	Jan. 12		
1901	THANE, B. L.....	Nov. 7	January*	38
1920	THOMPSON, ALBERT S.....	July 9		
1901	VEATCH, J. ALLEN.....	Jan. 3	March	154
1892	WALCOTT, CHARLES D.....	Feb. 9	March	153
1909	WARREN, GEORGE H.....	Jan. 23	April	194
1902	WAYNE, C.....	Oct. 7		
1913	WHITE, I. C.....	Nov. 25	March*	157
1880	WILLIAMS, DAVID.....	Oct. 29	December	525
1923	WOOD, CHARLES R.....	June 8	July	326
1921	YANDELL, LUNSFORD PITTS.....	Aug. 10	October	445
1887	ZEHNDER, C. H.....	Dec. 26	February*	88

The Latouche System of Mining as Developed at the Beatson Mine, Kennecott Copper Corporation, Latouche, Alaska

BY BE VAN PRESLEY,* LATOUCHE, ALASKA

(New York Meeting, February, 1928)

THERE has been developed at Latouche a rather unusual system of mining which, for want of a better name, has been called the "Latouche system of mining." It is a modified form of shrinkage stoping applicable to orebodies of large or fairly large size where the ground is such that timbering would not be required in stoping. Mention has been made of this method of mining by Birch,¹ who gave no details as it had been in use only a short time and the final outcome undetermined. The system, however, has proved to be entirely successful and it is thought that a paper covering it might be of general interest.

Before proceeding with a detailed description of the Latouche system a brief description of the property and the various mining methods previously used will be given.

LOCATION

The Beatson mine of the Kennecott Copper Corpn. is situated on the west coast of Latouche Island, in Prince William Sound, Southwestern Alaska, 45 miles east of Seward, the coast terminus of the U. S. Government Railroad. It is a regular port of call for ships on the Southwestern Alaska run. The island is about 12 miles long and 3 miles wide. A ridge of mountains runs the entire length with a maximum elevation of 2500 feet.

CLIMATE

Latouche has to contend with possibly the greatest amount of precipitation on the North American continent. The annual average precipitation for the past 4 years has been 188 in., with 248 in. for the year 1926. A rainfall of 4 in. per day is not uncommon. In the winter the thermometer rarely falls below +20° F. and the snowfall is usually heavy during the months of March and April.

* General Superintendent, Latouche plant, Kennecott Copper Corpn.

¹ S. Birch (President, Kennecott Copper Corpn., New York, N. Y.): Geology and Mining Methods of Beatson Mine. *Trans.* (1925) 72, 147.

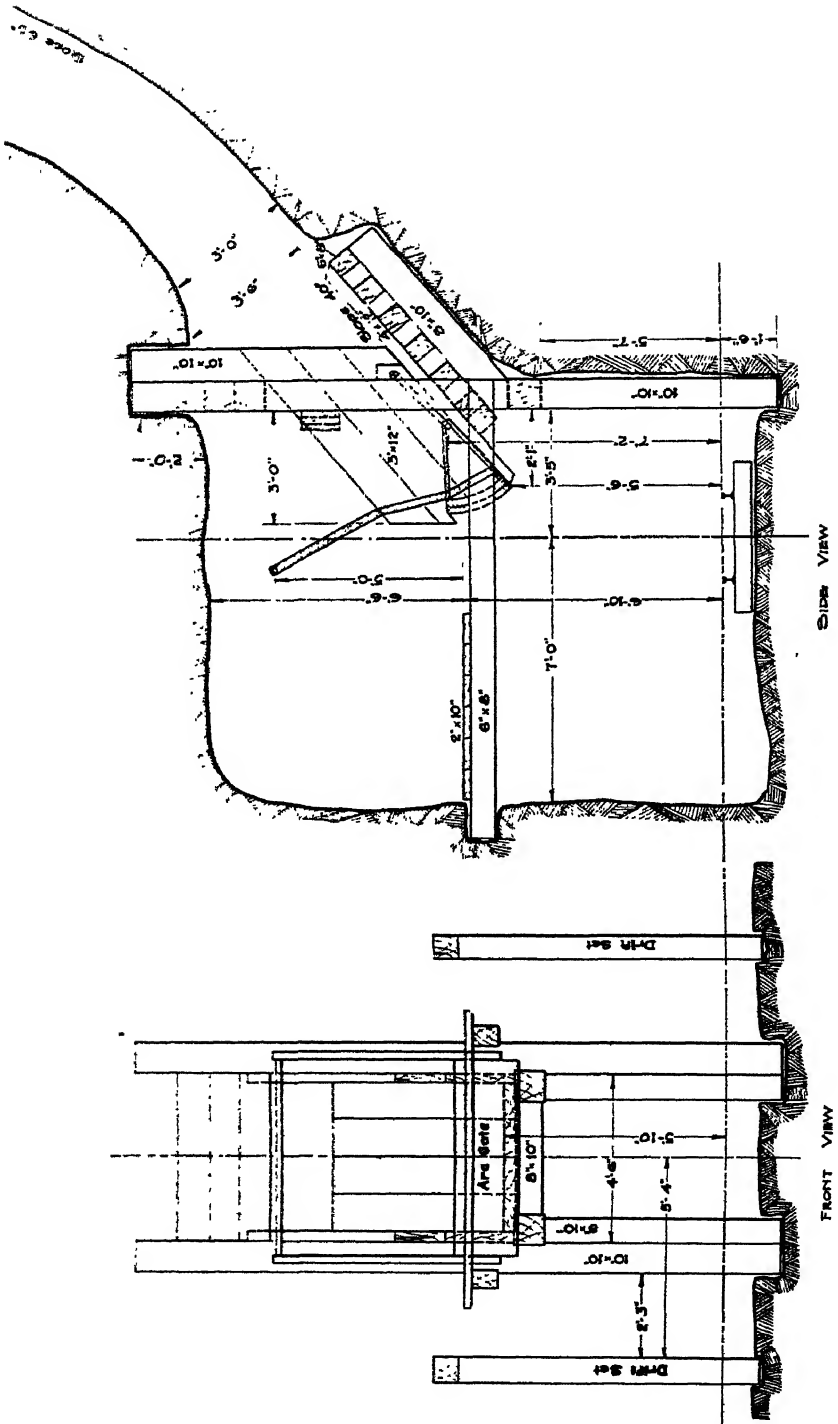


Fig. 1.—DETAIL OF CHUTES.

MINING METHODS

The mine has produced to Jan. 1, 1927, 4,078,000 tons of ore, with an additional 593,000 tons broken and remaining in the stopes on this date. The total tonnage produced from the various methods of mining is as follows:

	Tons
Bluff mining (glory-hole).....	2,161,272
Small shrinkage stopes.....	195,273
Square set stopes.....	34,000
Latouche method	1,687,455*
Total.....	4,078,000

* Includes ore from development.

Previous to 1923 most of the ore came from bluff or glory-hole mining, small shrinkage stopes above the bluff level and small square-set stopes in the higher grade ore along the hanging wall above the main level. Since January, 1923, practically all the ore produced has been derived from the Latouche system.

Square-set Stopes

There are no data available on the square-set mining. No doubt trouble was encountered, for in 1920 all these stopes were caved and filled with a mixture of ore and waste. They were not carried more than four floors in height, and about 30 ft. of ore remained between the back of the stopes and the surface. All this ore, however, is being reclaimed, the copper tenor being sufficiently great to carry the waste with which it is mixed.

Small Shrinkage Stopes

A group of small shrinkage stopes was opened up from the bluff level, the broken ore being transferred through long raises to the main level for tramming. In order to mine this ore by ordinary shrinkage-stope methods the stopes had to be carried small, due to the treacherous nature of the ground. As a result, their production was slow and the cost excessive.

Bluff or Glory-hole Mining

This method is essentially the same as is practiced elsewhere. Chute raises were driven from the main level to intermediate levels which varied in elevation above the main in different parts of the mine from 40 to 100 ft. A grizzly was installed on the intermediate level and a raise driven on through to the surface of the bluff. These raises, in some instances, were 100 ft. in length, above the grizzlies, and before the surrounding ore could be broken into them the raises had to be enlarged or

serious hang-ups would occur. The enlarging process was started in a raise about 10 ft. above the grizzly and the raise enlarged to 10 or 12 ft. in diameter. This was done by using a stoper drill and working up on the broken rock. The ground had such a tendency to cave, due to the numerous slips and schist belts as mentioned above, that even with an opening of 10 to 12 ft. in diameter the work was hazardous and many serious accidents resulted. To avoid the danger involved in the use of a stoper operated from the broken rock pile with unsafe ground overhead, a staging was placed in the raise and from this, using a jackhammer type of machine, flat holes were drilled into the walls of the raise. This solved the problem. Later, in order to drill longer holes, a small bench or station was cut in the raise and steel up to 15 ft. in length was used with more ground put on the holes. Eventually it was found that these raises could be carried up 30 ft. in diameter and that the vertical distance between the benches could be increased up to 20 feet.

This method of mining, by the enlargement of raises, proved to be very advantageous. The tonnage broken per machine drill shift or per labor shift employed was much higher than obtained in the regular glory-hole or open-pit mining. The powder consumption per ton of ore broken was also considerably less. It was, therefore, decided to utilize this scheme, if possible, in mining the ore below the main level which had to be extracted by some underground method.

The "Latouche mining method," described in detail below, was finally evolved.

"LATOUCHE MINING METHOD"—DETAILED DESCRIPTION

Blocking Out Stopes and Pillars

As the extent and the outline of the orebody on the main level was definitely known, alternate stopes and pillars were laid out in plan across the general strike of the ore, the width of each stope being 70 ft., and each pillar 30 ft. The length of the stopes and pillars varied from foot to hanging depending on the width of the orebody, the smallest stope being 70 ft. in length and the largest 290 ft. in length. (Fig. 2.)

Preliminary Development

The preliminary development work such as main haulage ways, pumping stations, sumps, ventilation system, etc., will not be touched upon in detail in this article as these follow general practice and present no features of unusual interest. The cost of all this work, however, is included in the ultimate cost of the ore derived from the system of mining outlined herein.

Stope Development

In order to prepare the stopes and pillars for mining operations, to carry on actual mining, bulldoze broken ore, and tram to shaft, under this

system of mining it is necessary to develop three levels: an upper level from which actual mining is carried on; an intermediate level from which

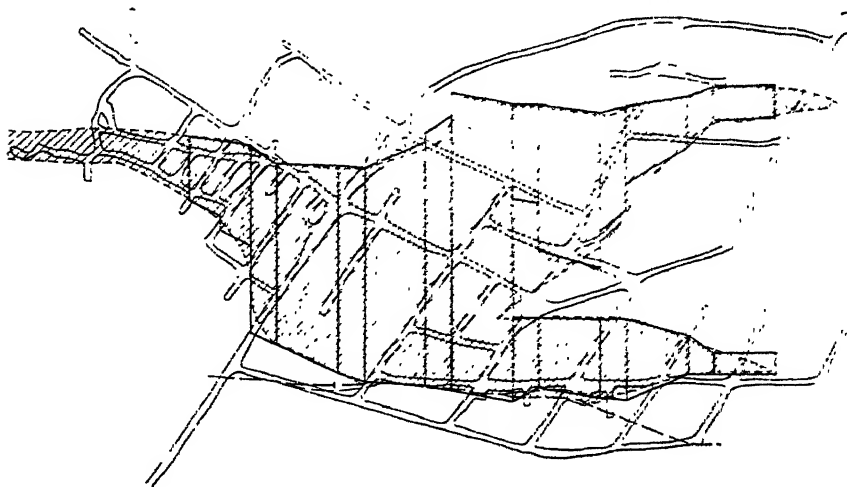


FIG. 2.—PLAN OF MAIN OR MINING LEVEL SHOWING OUTLINE OF OREBODY WITH ALTERNATE 70-FT. STOPE AND 30-FT. PILLARS LAID OUT ACROSS THE STRIKE. NO STOPE DEVELOPMENT WORK HAS BEEN DONE. WORKINGS SHOWN WERE USED IN CONNECTION WITH BLUFF MINING.

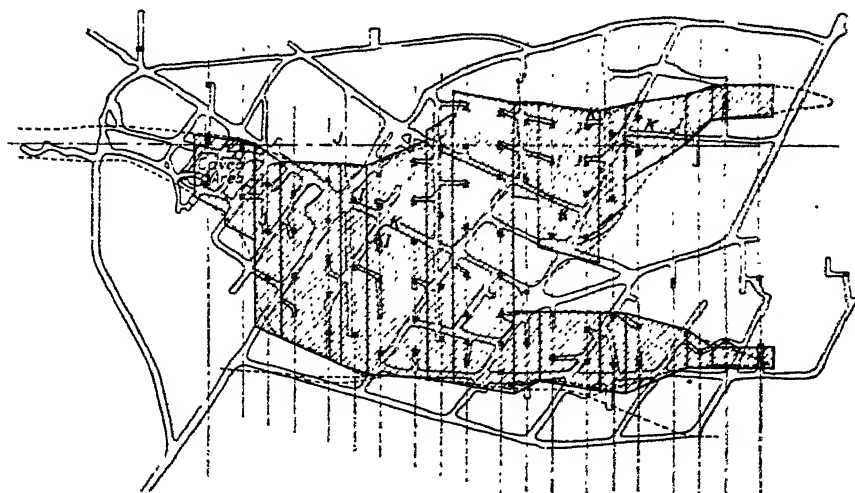


FIG. 3.—PLAN OF MAIN OR MINING LEVEL SHOWING OUTLINE OF STOPE AS MINED. ALSO STOPE DEVELOPMENT AND STOPE PREPARATION WORK AS COMPLETED. J. PILLAR CROSSCUTS. K. APPROACHES TO MINING RAISES. I. MINING RAISES.

the stopes are prepared and undercut and later used as a grizzly level; and a lower level for haulage. These will be referred to hereafter as the mining level, the grizzly level and the haulage level.

On the mining level 4 by 7 ft. crosscuts (*J*), are driven from the footwall drift closer to the hanging wall in each pillar, and later as the mining

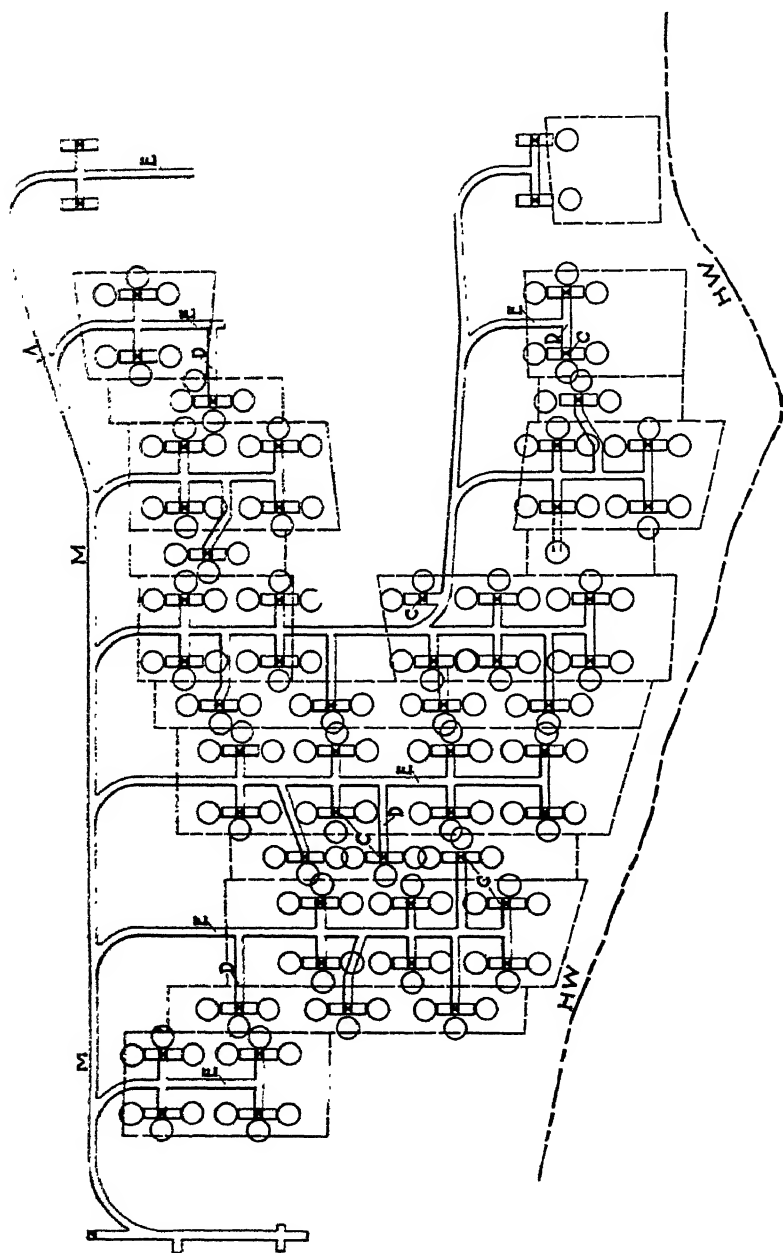


FIG. 4.—PLAN OF 150-FT. LEVEL OR GRIZZLY LEVEL, SHOWING VARIOUS WORKINGS WITH THE AREA DRAWN BY EACH GRIZZLY INDICATED BY CIRCLES. C. GRIZZLY CHAMBERS. D. APPROACHES TO GRIZZLIES. E. CROSSCUTS. M. FOOTWALL DRIFT. HW. TRACE OF HANGING WALL OR BEATSON FAULT ON 150-FT. LEVEL. DASH LINES ARE OUTLINES OF UNDERCUTS PROJECTED VERTICALLY TO THE PLANE OF THE 150-FT. LEVEL.

raises (*I*) are driven from the stopes to this level connections (*K*) are made from these pillar crosscuts to the raises. (Figs. 3 and 5.)

The grizzly level was laid out 150 ft. below the mining level. This level consists of a 5 by 7 ft. drift (M) driven well into the footwall, and 5 by 7 ft. crosscuts (E) every 100 ft., these crosscuts being laid out so that they run along the center line of each stope. Later on grizzly chambers (C) are cut on either side of these crosscuts. (Figs. 4 and 5.)

The haulage level was laid out 50 ft. below the grizzly level. A hanging wall drift and a footwall drift (N) 8 by 7½ ft. were driven, with 8 by 7½ ft. crosscuts (O) at 100-ft. intervals. The haulage cross-

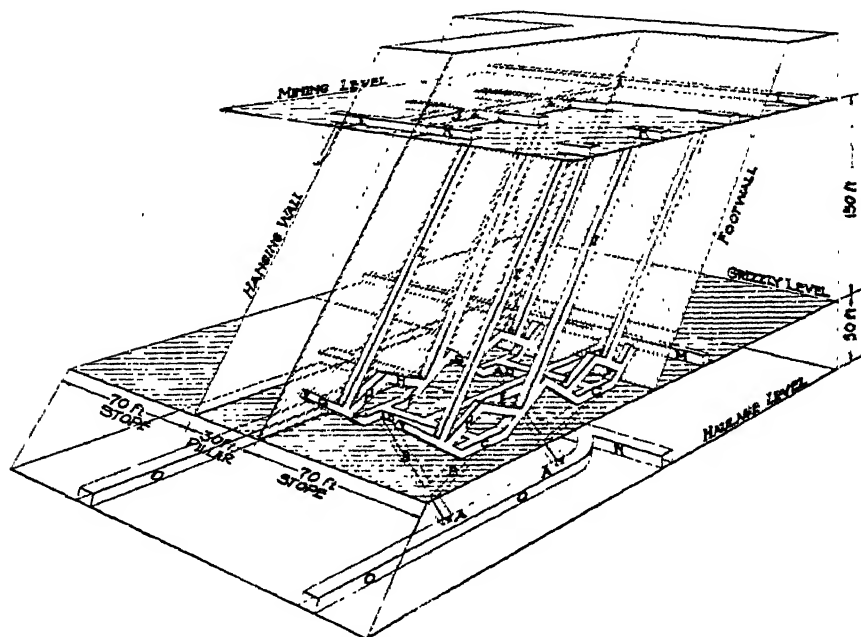


FIG. 5.—PHANTOM PERSPECTIVE OF A STOPE AND PILLAR SHOWING ALL STOPE DEVELOPMENT AND STOPE PREPARATION WORK COMPLETED.

- | | |
|--------------------------------|------------------------------------|
| A. Chutes. | H. Undercutting raises. |
| B. Forked chute raises. | I. Mining raises. |
| C. Grizzly chambers. | J. Pillar crosscuts mining level. |
| D. Grizzly chamber approaches. | K. Approaches to mining raises. |
| E. Crosscut on grizzly level. | L. Footwall drift mining level. |
| F. Raises to stope. | M. Footwall drift grizzly level. |
| G. Undercutting raises. | N. Footwall drift haulage level. |
| | O. Haulage crosscut haulage level. |

cuts were driven on the same bearing as the crosscuts on the grizzly level, but not vertically below, being offset 10 ft. to enable forked chute raises being driven later to serve two grizzlies on the level above. (See Figs. 5 and 6.)

After these three levels have been developed to this extent the ore is practically blocked out, and for convenience in laying out preparatory work vertical cross-sections normal to the strike of the ore are made, two for each stope and one for each pillar. For the stopes the sections are

Stope Preparation

After the stope development work has been completed, stope preparation work is laid out for each stope and pillar. The first step is to lay out the grizzly chambers (C). These are 24 ft. long by about 8 ft. wide and $7\frac{1}{2}$ ft. high. The longitudinal center line of the stope grizzlies is 15 ft. in from the pillar line on either side of the stope or 20 ft. on either side of the center of the crosscuts on the grizzly level. The pillar grizzlies (X) are along the center line of the pillar or 50 ft. from the center line of the grizzly level crosscuts. In the stopes there are two lines of grizzlies; in the pillars, one line. These grizzlies are spaced from footwall to hanging wall at intervals of from 60 to 75 ft. center to center, although in some instances it is necessary to space them closer together to serve properly the stope or pillar. The grizzly nearest the footwall is laid out so that its footwall point of draw coincides with the footwall or slightly into the footwall of the stope. As the orebody dips at 60° to 70° the hanging wall grizzlies are located 70 to 80 ft. from the hanging wall of the ore. (See plan of grizzly level, Fig. 4.)

The chutes (A) are then laid out on the haulage level; one chute serving two stope grizzlies, one on either side of the crosscut above. The pillar chutes are located on the opposite side of the haulage crosscuts and serve only one grizzly. (See Fig. 6.)

In driving the work outlined above, the usual procedure is to cut out for and install the chutes on the haulage level and drive the chute raises to the grizzly level above. While this is being done the approaches (D) to the grizzly chambers on grizzly level are driven. After the chute raises have holed through to the grizzly approaches, the grizzly chambers are cut. The chambers could be cut previous to this, but if delayed until the chute raises are through considerable expense is saved in disposing of the muck from the grizzly chamber cutouts.

The grizzlies themselves may now be installed, although this is not essential at this time and may be postponed until the stope is ready for undercutting. In any event the chute raise is belled out to the extent required for a grizzly. (See grizzly details, Fig. 7.)

From the end of each grizzly chamber raises are driven. These raises are driven at an angle of 40° with the horizontal and their center line lies in a plane parallel with the pillar and 15 ft. from it. The first 12 ft. of these raises are driven $4\frac{1}{2}$ by $4\frac{1}{2}$ ft. in section and thereafter 5 by 5 ft. The smaller section raise (F) is called the "raise to stope" (as this is not undercut later), and the remaining portion "undercutting raise" (G). In driving these raises the hanging wall raise from one grizzly will connect with the footwall raise of the grizzly chamber next to it, and all the grizzly chambers in one line thus connected. The undercutting raises on the footwall are driven a few feet past the ore limits while those on the hanging wall are stoped when the hanging wall is reached.

After these undercutting raises are driven another series of undercutting raises (*H*) is driven back over the grizzly chambers. These raises are started from the first raises driven (at a point 12 to 15 ft. up) at an angle of 40° with the horizontal and with their center line in the same vertical plane as the first raises. These raises meet in an apex vertically above the center of each grizzly chamber.

After the undercutting raises are completed the "mining raises" (*I*) are driven. These are 5 by 5 ft. raises making an angle of from 55° to 70° with the horizontal depending upon the dip of the ore in the particu-

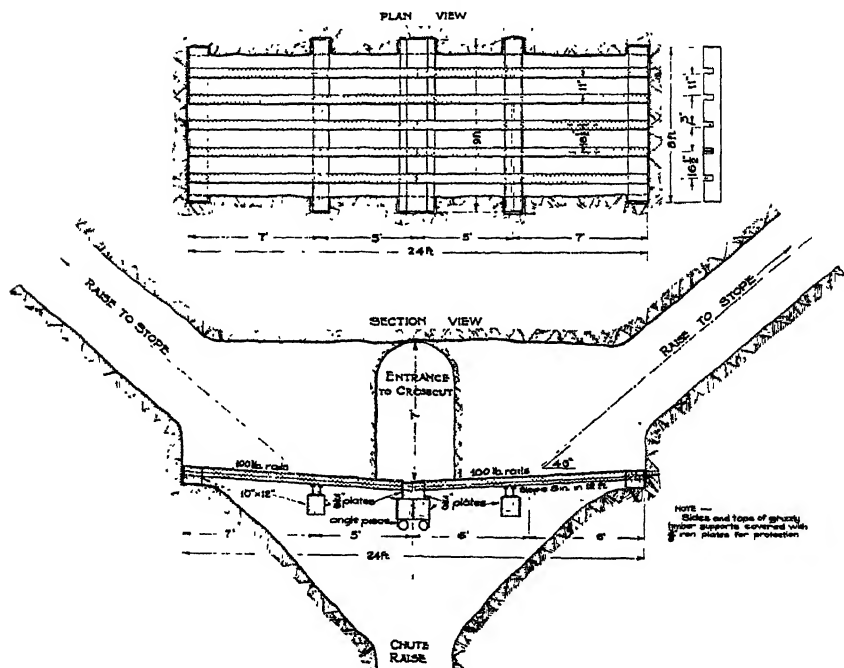


FIG. 7.—GRIZZLY CHAMBER SHOWING DETAILS OF GRIZZLY. A THIRD POINT OF DRAW, NOT SHOWN IN THE DRAWING, IS INSTALLED AFTER THE STOPE IS MINED. THIS IS DONE BY DRIVING A RAISE TO THE STOPE FLOOR FROM THE SIDE OF THE GRIZZLY CHAMBER OPPOSITE THE "ENTRANCE TO CROSSCUT." THIS RAISE LIES IN A VERTICAL PLANE AT RIGHT ANGLES WITH THE PLANE OF THE SECTION SHOWN IN SKETCH.

lar stope or pillar. In the stopes the center lines of these raises are in the plane referred to above—15 ft. in and parallel with the pillar. In the pillars, the center lines of the mining raises lie in a vertical plane through the center of the pillar. The spacing of these raises from foot to hanging of the orebody is from 40 to 60 ft. center to center measured on the horizontal. The raises nearest the hanging wall are usually from 15 to 25 ft. in from the wall, while those on the foot are somewhat closer, 10 to 15 ft. and in some instances right along the footwall.

While the mining raises are being driven, approaches to them (*K*) are driven on the mining level. After the mining raises are connected

up with the mining level each raise is thoroughly barred down and ordinary wooden ladders made of rough 2 by 4 in. material with 1 by 4 in. treads are installed. As the mining raises are all driven accurately with respect to line and grade, the ladders can be put in without twists or bends. After the ladders are in place a 1 by 3 in. batten is nailed on top of the 2 by 4 in. piece on each side of the ladder. These act as guides for the small skip which is used in each raise for handling tools and supplies. As soon as the raise is to be used for either undercutting or mining a tugger hoist is installed either on a bar or on a timber at the top of the raise.

The stope now has been completely prepared as shown in the phantom perspective, Fig. 5. There yet remains the undercutting before the stope is ready to be mined.

Undercutting

In the stopes, undercutting is accomplished by widening out laterally the undercutting raises from each grizzly until these break through in the center of the stope to the corresponding undercut from the raises on the grizzly opposite. On the pillar sides of the raises the widening out is carried on until the pillar line is reached. Undercutting is done with CC 11 stope machines using 1 in. quarter octagon steel. Usually a temporary pillar, about 10 ft. through, running from apex to apex above each pair of opposite grizzlies is left. These pillars are drilled but not blasted until the remainder of the stope is completely undercut. They are blasted just previous to the start of mining operations and access to them is gained by coming down through the mining raises. This has been found to be a safer procedure than to attempt to undercut entirely a stope without leaving pillars. The undercutting operations are carried on from the grizzly level, one side of the grizzly always being kept open for access to the work. If for any reason, such as a large fall of ground from the back, this opening should be closed, the mining raises afford a means of exit to the level above.

In undercutting the pillars, which is not done until the stope on each side has been completely mined, the undercutting raises are widened out 10 ft. on either side of the center line. This leaves a 5-ft. longitudinal pillar between the broken ore in the stope and the pillar undercut. Usually these pillars are neither drilled nor blasted for when mining starts the first round above the undercut in the pillar breaks through to the stope on either side and the stump of the pillar left serves to keep the pillar grizzlies from drawing ore from the stopes.

The foregoing outlines the work necessary before the stopes and pillars are in readiness to mine. Table 1 gives the cost of stope preparation, pillar preparation, stope development and preliminary development for opening up the entire orebody above the stope undercuts containing

2,603,259 tons of ore (estimated). This table shows the number of feet driven, average cost per foot, number of units (chutes and grizzlies), the average cost each, and the total amount expended for each item. Based on the estimated tonnage above the stope undercuts, this amounts to 29 c. per ton of ore. This is given in greater detail in Appendix A which gives each stope and pillar in detail.

TABLE 1.—*Cost of Stope Preparation, Pillar Preparation, Stope Development and Preliminary Development for Opening Up an Orebody Estimated to Contain 2,603,259 Tons*

	Total Feet	Total Number	Total Amount	Per Foot	Cost Each
Stope preparation:					
Chutes.....		22	\$ 21,396.58		\$972.57
Chute raises.....	2,111		18,466.49	\$ 8.75	
Grizzly cutout.....	950		11,826.41	12.45	
Grizzly installation.....		42	24,979.00		594.74
Grizzly level.....	3,968		53,406.61	13.46	
Raises to stope.....	1,732		15,371.97	8.88	
Mining raises.....	11,131		93,263.63	8.38	
Undercutting raises.....	3,992		27,815.71	6.97	
Undercutting (sq. ft.).....	120,074		84,584.16	0.704	
Drifts and crosscuts.....	1,991		21,635.06	10.87	
Timbering.....			8,555.67		
Total.....			381,301.29		
Pillar preparation:					
Chutes.....		13	6,499.88		499.99
Chute raises.....	791		6,623.85	8.37	
Grizzly raises.....	25		182.43	7.30	
Grizzly cutout.....	270		3,754.32	13.91	
Grizzly installation.....		15	6,622.86		441.52
Raises to stope.....	280		2,544.18	9.09	
Pillar mining raises.....	2,896		22,649.78	7.82	
Undercutting raises.....	2,089		15,142.03	7.25	
Undercutting (sq. ft.).....	28,871		18,729.76	0.649	
Drifts and crosscuts.....	2,060		22,559.00	10.97	
Timbering.....			3,318.13		
Prospect raises.....	47		448.63	9.55	
Total.....			109,104.85		
Prospect raises and crosscuts.....	907		11,473.71	12.65	
Ventilation raises.....	323		4,524.31	14.01	
Ventilation crosscut.....	262		2,648.29	10.11	
Stope development:					
Footwall drift.....	624		18,240.97	29.23	
South drift.....	194		6,200.43	31.96	
Crosscuts.....	2,882		62,038.91	21.53	
S. H. W. drift.....	562		13,888.70	24.71	
Miscellaneous drifts.....	981		18,862.48	19.23	
Total.....			119,231.49		
Preliminary development:					
Sump and drainage.....			41,453.92		
New sinking hoist.....			5,565.33		
Ore pocket.....			24,880.00		
Main entry.....			18,979.27		
Aerial tram.....			2,210.60		
201 manway.....			11,338.45		
Total.....			104,427.57		
Diamond drilling.....			22,512.21		
Grand total.....			\$755,223.72*		

* Based on estimated tonnage this amounts to \$0.29 per ton.

COSTS

In connection with the foregoing costs it might be well at the outset to explain the Latouche system of cost-keeping. The cost on each individual working place is kept separately and to it charged not only the direct costs, but all indirect costs and its proper proportion of general mine charges. For instance, a raise being driven is charged with the actual cost of driving, such as labor and supplies; a proportion of machine drill expense which includes compressor operation, steel sharpening, repairs, etc.; cost of tramming and hoisting the ore or waste broken; and a proportion of the general mine expense which includes supervision, ventilation, drainage, etc. The cost then as shown for any particular working is the total cost. There are no undistributed mine charges. The mine work is divided into the following classifications: (1) Exploration, (2) development, (3) stope preparation, (4) ore production. The sum of the amounts spent on various exploration jobs would give the entire amount expended on exploration; the sum of the amounts spent on the various development jobs would give the entire amount spent on development and so on. The sum of the amounts spent for exploration, development, stope preparation and ore production, is the total mine cost.

Ore production is split into three divisions: (1) "Breaking ground," (2) "ore through grizzlies," and (3) "loading, tramming and hoisting." Each division is charged with its proper direct charges and a proportion of the general mine expense, so that for any operation the total cost is shown.

The costs given in this article are complete from the time of starting work on the "Latouche system of mining" until Jan. 1, 1927. On the latter date practically all development, stope preparation and undercutting work had been completed for the entire orebody above the levels shown. The costs for these classes of work and the cost per ton of ore developed may be accepted as final. At the present writing mining in all the stopes and pillars has been completed and the ultimate cost of breaking ground (mining in stopes and pillars) per ton will be slightly less than shown herein as a portion of the ore which has caved above the mining level has not been credited as "tons broken."

ORE PRODUCTION

Mining Operations in Stopes

After a stope is completely undercut, with the exception of the small pillars left across the grizzly apexes, the back of the stope is a series of more or less regular rills. If the rills are very solid it is necessary to drill them, more particularly in the footwall portion of the stope where mining operations are always started. To drill these rills, Bull Moose machines

are used and holes up to 20 ft. in length put in, with a slight downward slope, flat or even with a slight slope upward, as shown in Fig. 8 by holes numbered 1. These holes are blasted at the same time that the small

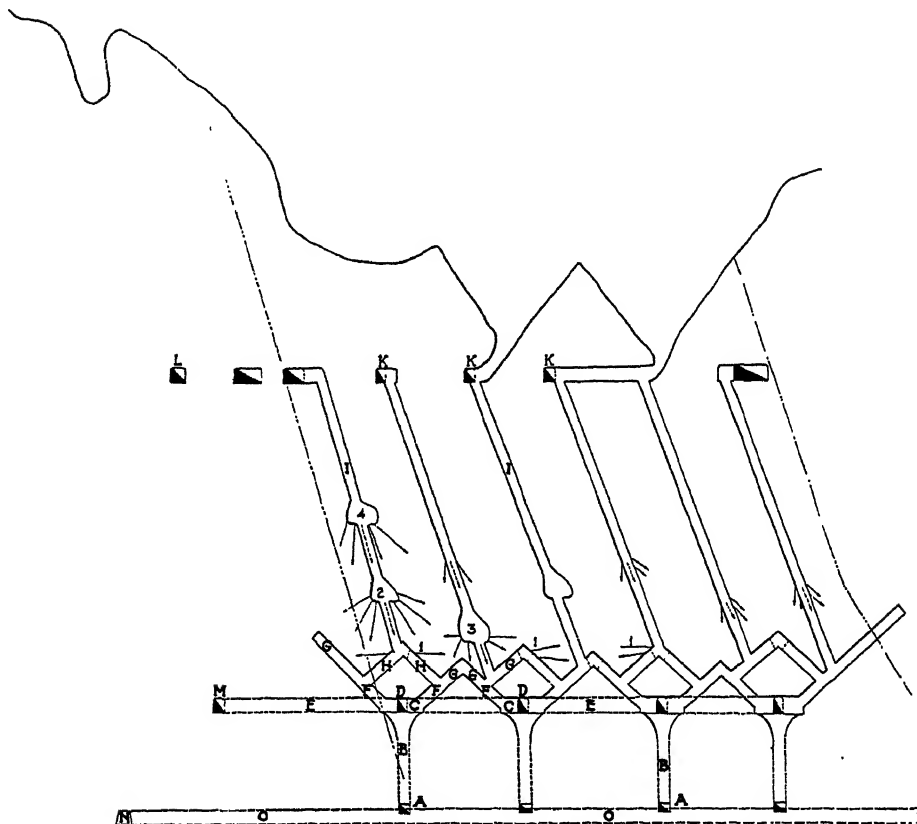


FIG. 8.—NORTH SECTION OF 202 STOPE, ALL PREPARATION WORK COMPLETED AND STOPE ENTIRELY UNDERCUT. ROUNDS DRILLED DURING THE FIRST 2 MONTHS MINING SHOWN. NUMBERS INDICATE ORDER OF BLASTING THE ROUNDS. HOLES FOR ENLARGING MINING RAISES PREPARING FOR SUCCEEDING ROUNDS ALSO SHOWN. DOTTED LINES OVER APEXES INDICATE PILLARS LEFT WHILE UNDERCUTTING. THESE ARE BLASTED AT THE SAME TIME AS HOLE No. 1.

A. Chutes.

B. Forked chute raises.

C. Grizzly chambers.

D. Grizzly chamber approaches.

E. Crosscut on grizzly level.

F. Raises to stope.

G. Undercutting raises.

H. Undercutting raises.

I. Mining raises.

K. Approaches to mining raises.

L. Footwall drift mining level.

M. Footwall drift grizzly level.

N. Footwall drift haulage level.

O. Haulage crosscut haulage level.

undercut pillars are blasted, or if the ground is particularly solid the undercut pillars are blasted after the holes in the rills (No. 1) have been drilled, and the holes in the rills blasted a day or two later. The stope is now in condition to start mining from the mining raises.

Mining is started in the footwall raises. Coming down from the mining level to a point 30 to 40 ft. above the back of the stope a temporary staging is installed in the mining raise and holes drilled downward in the sides, back and foot to a depth of 8 to 10 ft. These holes are fanned out so that when blasted the raise is enlarged in all directions and a bench formed. It is often necessary to take two swipes before a bench of sufficient size is made. These benches are from 18 to 25 ft. long, and from

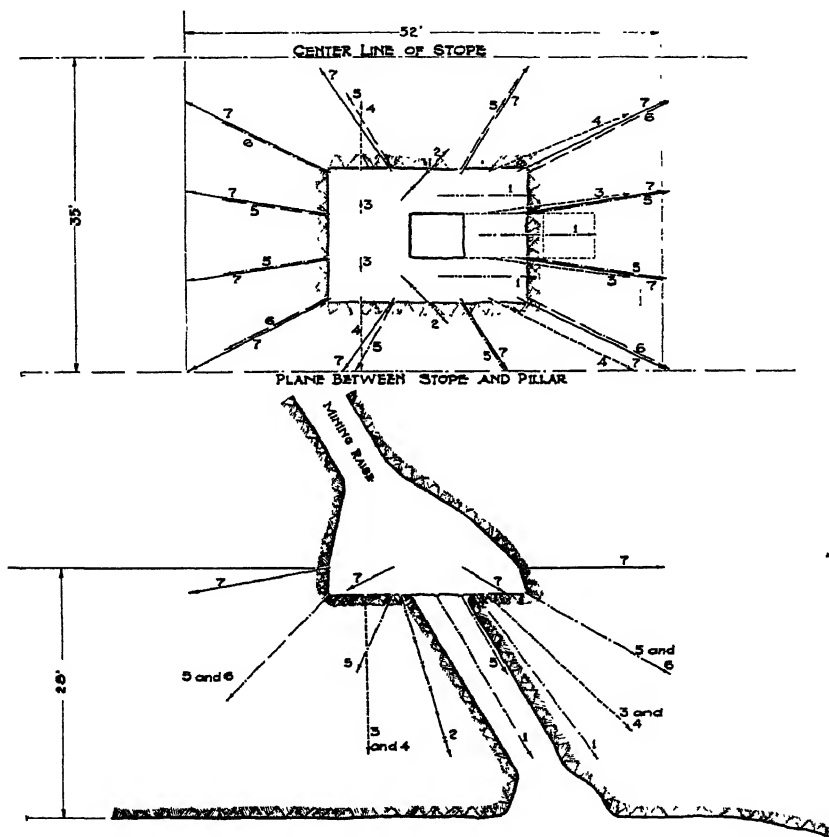


FIG. 9.—PLAN AND VERTICAL SECTION SHOWING TYPICAL LONG ROUND USED IN HARD GROUND. NUMBERS SHOWN ON HOLES INDICATE ORDER OF BLASTING.

12 to 14 ft. wide with the mining raise coming up through the center. This work is done with either a Bull Moose or Jackhammer machine and is designated as the "short round." The floor of this bench then is from 20 to 30 ft. above the back of the stope.

As soon as the short round bench has been cleared of muck, shoveling and barring down loose requiring from 4 to 8 hr. time for two miners, a staging is placed over the raise opening and drilling of the "long round" commenced. There is no absolute standard for the long rounds, although

they are all very similar, the holes being drilled to conform to the ground. If the ground is very hard from 35 to 40 holes are necessary while if the ground is softer and somewhat cracked up only 20 to 26 holes are needed. Fig. 9 shows a typical long round used in hard ground. Fig. 10 is an isometric drawing of the same round from which a clearer conception may be obtained. The holes in the long round are all drilled with Ingersoll-Rand Bull Moose machines using $1\frac{1}{8}$ -in. hollow hexagon steel having a 4-point cross bit. Steel up to 24 ft. in length is used, with changes in gage every 4 ft. from $2\frac{1}{8}$ -in. starters down to $1\frac{1}{4}$ -in. on the longest steel.

After the round has been drilled all the holes are sprung from one to three times to form a large pocket at the bottom of the hole for the powder used in blasting. On the first springing two sticks of $1\frac{1}{8}$ by 8 in. powder are used, on the second springing four sticks and on the third six sticks—in extremely hard ground nine or ten sticks per hole have been used. Springing a large round requires about a day for two miners.

The long round having been sprung, all holes are thoroughly cleaned out with a compressed air blowpipe. The round is then loaded with 40 per cent. Red Cross powder. No. 8 electric blasting caps are used hooked up in parallel series. From six to eight delays are used depending on the number of holes in a round. Fig. 9 shows the order in which the holes are blasted. Current is obtained from either a large size blasting machine or from a 110-volt lighting circuit. Although not absolutely necessary, the blasting of all the larger rounds is done after regular hours when most of the crew are out of the mine.

The amount used per round will vary from 500 to 1000 lb., depending on the number of holes in the round and the nature of the ground. Including the powder used in blasting the short rounds, springing holes and blasting the long rounds, 396,091 lb. were used in blasting 379 complete rounds or an average of 1045 lb. per round and amounting to 0.245 lb. of powder per ton of ore broken. These 379 rounds broke a total of 1,618,212 tons or an average of 4275 tons per round. The round shown in Fig. 9 would break a block of ground 52 ft. long by 35 ft. wide by 28 ft. thick, or a volume of 50,960 cu. ft. of rock in place which at 12 cu. ft. to the ton amounts to something over 4200 tons.

Two miners only are used on each crew in each mining raise, the work in any particular raise being done on one shift only when possible. The objections to working two crews on opposite shifts in the same raise are many, although it can be done. The principal reason for not doubling up on the crews is that the miners are paid a bonus based on the footage drilled. To increase the efficiency and insure no lost time to the miners one crew is often given two raises in which to work. The cycle of operations in connection with a complete round based on a crew of two miners working one shift is approximately as follows:

TYPE OF WORK	DAYS LAPSED TIME	MAN SHIFTS (8 Hr.)
Staging up for short round.....	$\frac{1}{4}$	$\frac{1}{2}$
Drilling short round.....	2 to 3	4 to 6
Springing short round.....	$\frac{1}{4}$	$\frac{1}{2}$
Blasting short round.....	$\frac{1}{2}$	1
Cleaning off bench.....	$\frac{1}{2}$	1
Barring down.....	$\frac{1}{2}$	1
Staging up for long round.....	1	2
Drilling long round.....	7 to 9	14 to 18
Springing long round.....	1	2
Loading and blasting long round.....	1	2
	14 to 17	28 to 34

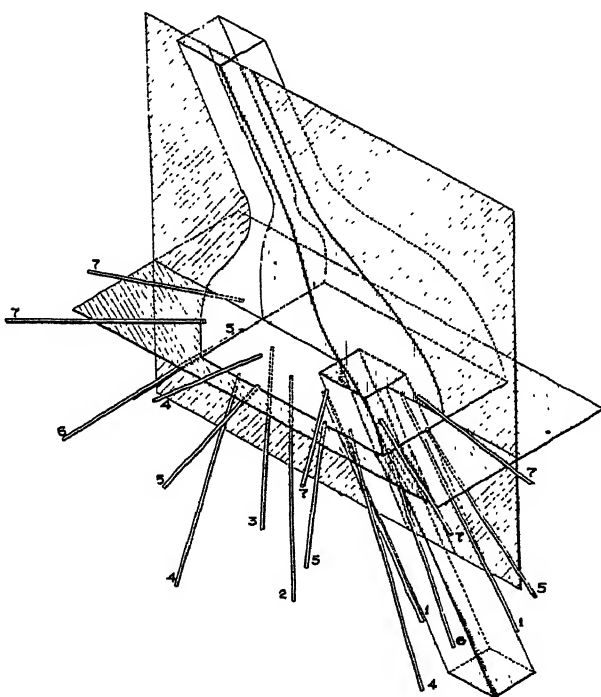


FIG. 10.—ISOMETRIC OF LONG ROUND SHOWN IN FIG. 9. NUMBERS SHOWN ON HOLES INDICATE ORDER OF BLASTING.

In breaking the total tonnage mentioned above, 8158 machine-drill shifts were required; this gives an average of 198 tons per machine-drill shift.

Complete technical data, in detail, covering the mining operations in the stopes are shown in Appendix G.

The cycle of operations is repeated in each mining raise until it is mined through to the mining level. It was found, however, in the Beatson mine, that the way the back of the stope is carried is very important. In another mine where conditions were not similar this would not be so important. In the Beatson mine, the ore near the hanging wall

is very hard to within from 2 to 5 ft. of hanging, there is then a band of soft ore and the gouge along the fault which forms the hanging wall. The country rock in the hanging wall is a soft short slate. The soft ore, the gouge, and the soft slate all cave very readily and very rapidly so that if any large portion of the hanging wall is exposed it caves and the ore in the stope becomes diluted with waste. This dilution is increased

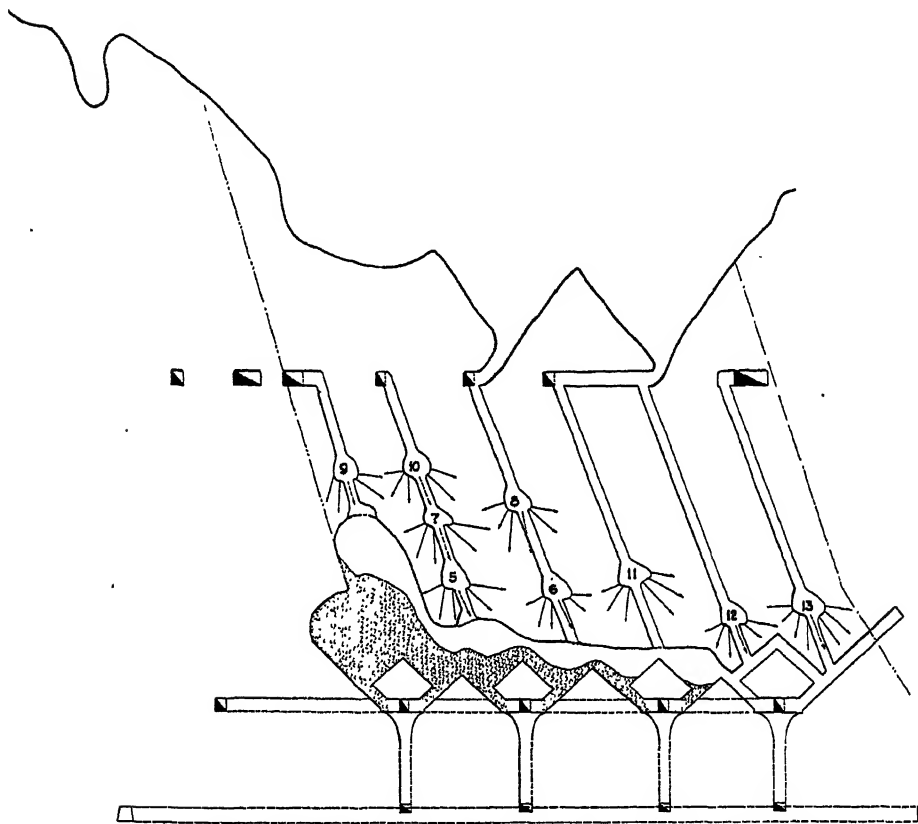


FIG. 11.—NORTH SECTION OF 202 STOPE, AT THE END OF 2 MONTHS' MINING SHOWING OUTLINE OF BACK AND ROUNDS TO BE DRILLED AND BLASTED DURING THE THIRD AND FOURTH MONTHS. NUMBERS ON ROUNDS INDICATE ORDER OF BLASTING.

when the stope is drawn to make room for blasting. In one instance during the early stages in the use of this system a small stope was lost entirely by being mined too far in advance in the hanging and drawing the stope too low. Two measures were used to overcome this tendency of the hanging wall to cave: one was to mine not quite through the hard ore and leave a thin shell of harder material along the hanging, the other was to carry the stopes up in such a manner that the footwall portion was well in advance of the hanging wall portion. This is clearly illus-

trated in Figs. 8, 11, 12 and 13 which are vertical sections through 202 stope showing the mining of this stope in successive stages. (Fig. 14 gives a recapitulation of all rounds drilled.) It will be noted that the back of the stope slopes down from foot to hanging until the mining in the footwall raises has been almost completed. It is then necessary to carry up the hanging wall portion and retreat toward the foot and finish

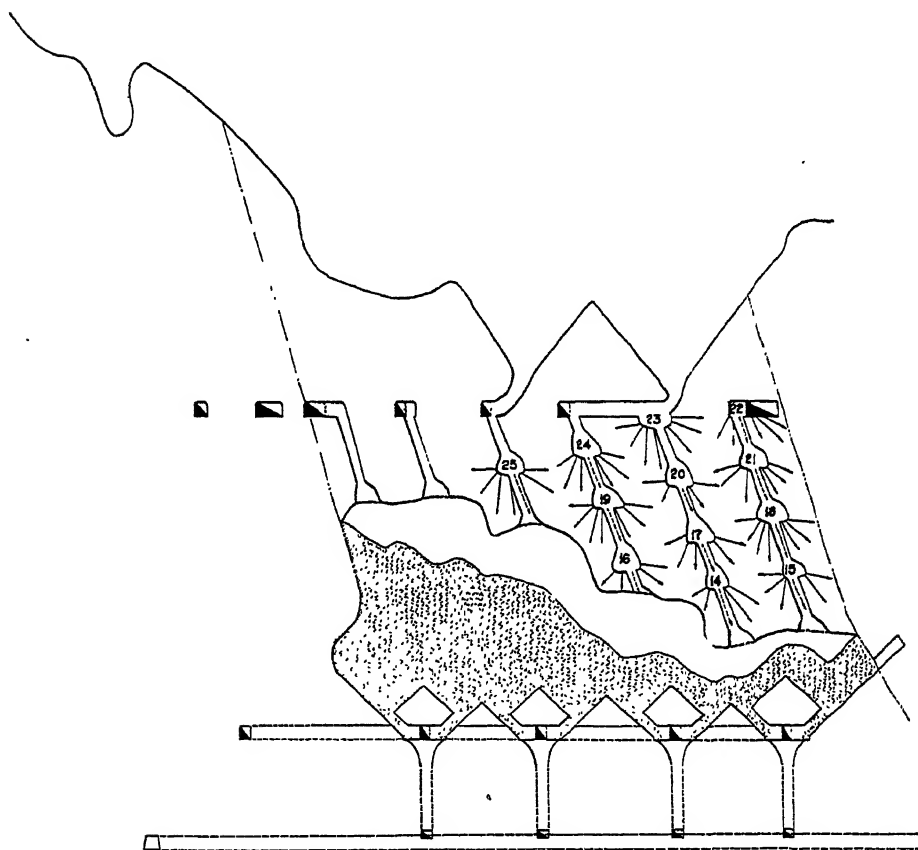


FIG. 12.—NORTH SECTION OF 202 STOPE AT THE END OF 4 MONTHS' MINING, SHOWING OUTLINE OF BACK AND ROUNDS TO BE DRILLED AND BLASTED DURING THE FIFTH AND SIXTH MONTHS. NUMBERS ON ROUNDS INDICATE ORDER OF BLASTING.

up in the footwall raises. All during the mining operations, in any of the stopes whose hanging wall coincided with the hanging wall fault, the broken ore was always kept up against the hanging. This was done by not drawing the hanging wall grizzlies any more than enough to relieve the stope in the early stages of mining in the hanging wall portion.

The cost of breaking ore in the stopes is shown in Table 2 below and in greater detail for each stope in Appendix B.

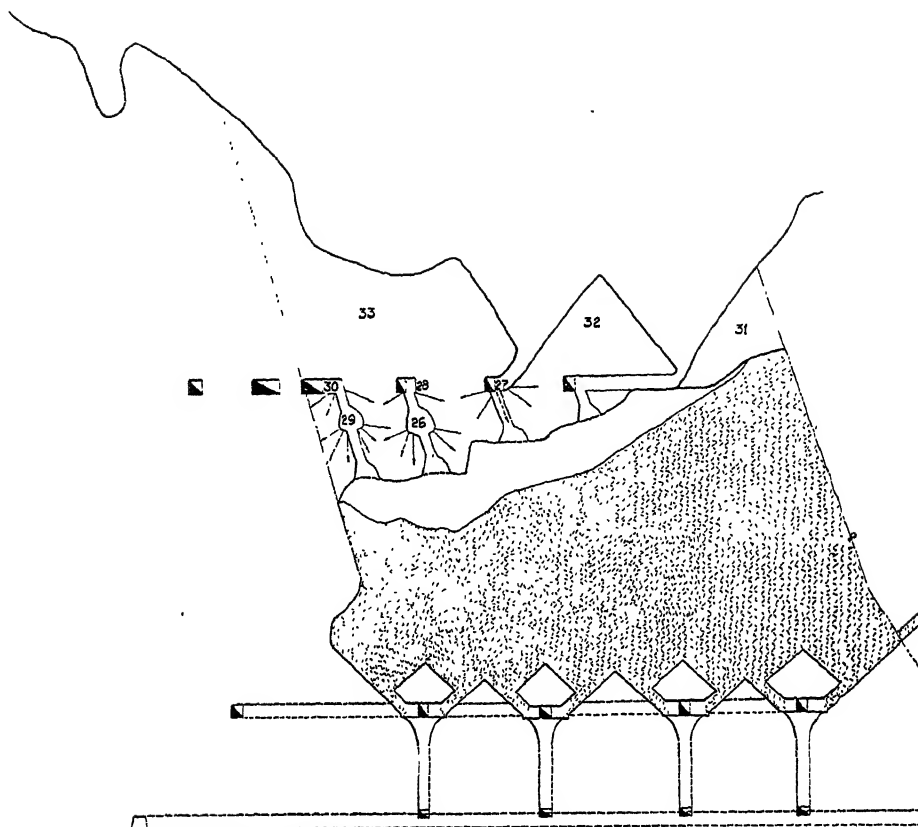


FIG. 13.—NORTH SECTION OF 202 STOPE AT END OF 6 MONTHS SHOWING OUTLINE OF BACK AND ROUNDS TO BE DRILLED AND BLASTED DURING SEVENTH AND EIGHTH MONTHS. NUMBERS ON ROUNDS INDICATE ORDER OF BLASTING.

Note on rounds numbered 31, 32, 33: Area above main level which did not cave was benched in by holes along pillars.

TABLE 2.—*Cost of Breaking Ground in Stopes*

[Tons Broken, 1,618,212]

	AMOUNT	PER TON
Miners and helpers.....	\$ 87,108.88	\$0.054
Timber supplies.....	3,380.35	0.002
Explosives.....	63,886.90	0.039
*Machine drill expense.....	63,350.11	0.039
†Miscellaneous and general mine expense.....	43,327.99	0.027
Total.....	\$261,054.23	\$0.161

* Machine drill expense includes cost of steel, steel sharpening, repairs to drills, compressor operation, installation of air and water lines, etc. It is distributed over the various jobs on the basis of the number of machine drill shifts employed.

† Miscellaneous charges include items not shown under other headings and are direct charges. General mine expense includes cost of supervision, ventilation, drainage and general mine supplies, and is distributed on the basis of the number of labor shifts employed.

Mining Operations in Pillars

The mining operations in the pillars are carried on in exactly the same manner as in the stopes with the exception that the back is carried hori-

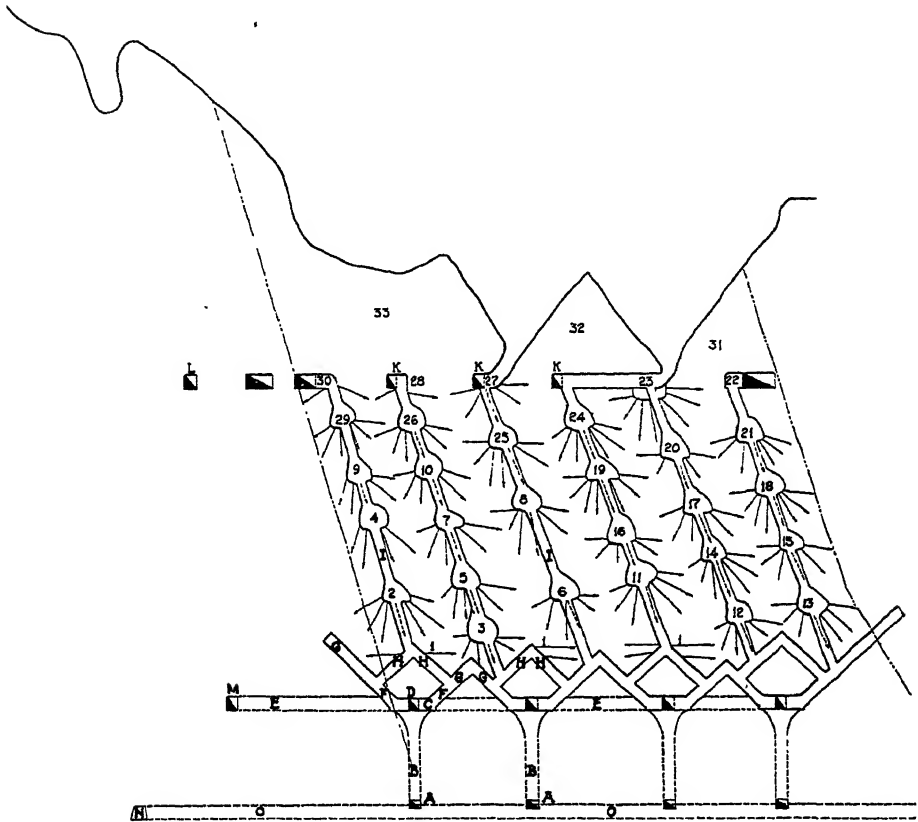


FIG. 14.—NORTH SECTION OF 202 STOPE GIVING A RECAPITULATION OF ALL ROUNDS DRILLED.

- | | |
|--------------------------------|------------------------------------|
| A. Chutes. | H. Undercutting raises. |
| B. Forked chute raises. | I. Mining raises. |
| C. Grizzly chambers. | K. Approaches to mining raises. |
| D. Grizzly chamber approaches. | L. Footwall drift mining level. |
| E. Crosscut on grizzly level. | M. Footwall drift grizzly level. |
| F. Raises to stope. | N. Footwall drift haulage level. |
| G. Undercutting raises. | O. Haulage crosscut haulage level. |

NOTE.—The south section of this stope was carried up simultaneously with an equal number of rounds. For technical data and costs on this stope see Tables *G* and *B* appended.

zonally during the time of mining. Mining is started in the pillar as soon as the stopes on both sides have been completely mined. The pillar preparation as mentioned, is usually done while mining in the adjacent stopes is being carried on. In mining the longer pillars it was found that

not more than two long rounds could be drilled and blasted before the pillar caved. In the shorter pillars mining was carried clear through to the mining level, the ground above here caving. While mining in the pillars it is necessary to discontinue drawing ore from the line of grizzlies adjacent to the pillar in the stope on either side. If these are drawn the pillar is apt to settle, interfere with the pillar mining and make the work hazardous. The longer pillars would probably cave after the first round in each mining raise in the pillar had been blasted, if the pillar

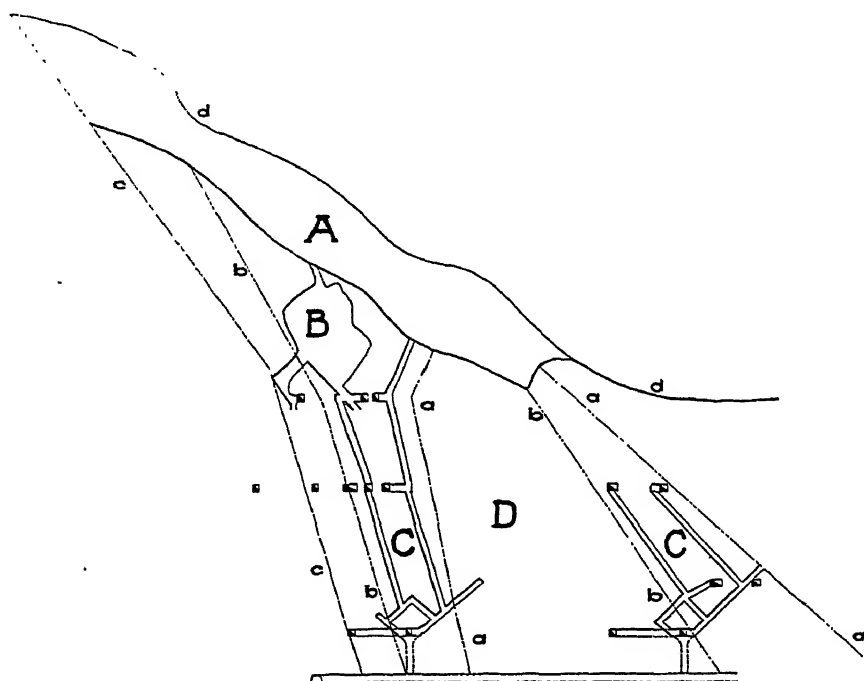


FIG. 15.—VERTICAL SECTION NORMAL TO STRIKE THROUGH SOUTH END OF OREBODY, SHOWING RELATIVE AREAS MINED BY VARIOUS METHODS.

- A. Bluff mining.
- B. Small shrinkage stope.
- C. Latouche system.
- D. Horse of waste.

- a. Hanging wall.
- b. Economic footwall.
- c. Limit of mineralization.
- d. Original surface.

grizzlies and the stope on either side were drawn heavily. But if forced to cave at this early stage it would cave in large blocks making the drawing of the ore through the pillar grizzlies very difficult and expensive. For this reason it is better to hold off caving and mine in the pillar as long as possible. At the date of this writing all pillars have been mined without a serious accident.

The cost of breaking ore in the pillars is shown in Table 3 below and in greater detail in Appendix B. Technical data covering the mining in pillars are also shown in Appendix G.

TABLE 3.—*Cost of Breaking Ground in Pillars*

[Tons Broken, 501,741]

	AMOUNT	PER TON
Miners and helpers.....	\$22,762.77	\$0.045
Timber supplies.....	1,141.43	0.002
Explosives.....	14,895.12	0.031
Machine drill expense.....	15,153.04	0.030
Miscellaneous and general mine expense.....	14,534.54	0.029
Total.....	\$68,491.90	\$0.137

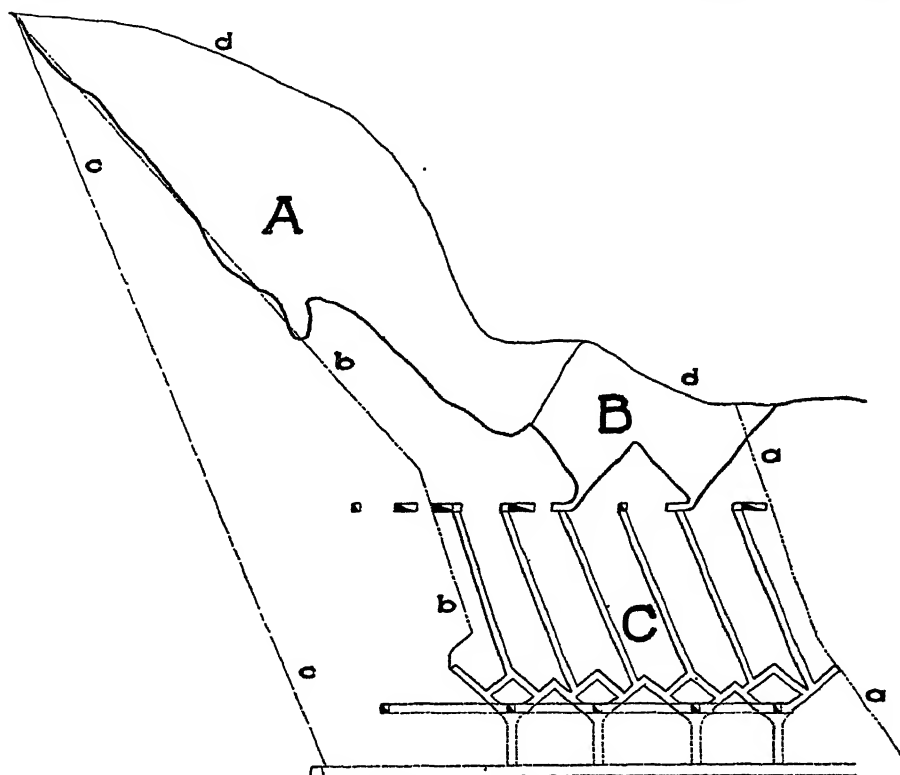


FIG. 16.—VERTICAL SECTION, NORMAL TO STRIKE, THROUGH CENTRAL PART OF OREBODY (NORTH SECTION OF 202 STOPE), SHOWING RELATIVE AREAS MINED BY VARIOUS METHODS.

A. Bluff mining.
 B. Benchng.
 C. Latouche system.

a. Hanging wall—Beatson fault.]
 b. Economic footwall.
 c. Limit of mineralization.
 d. Original surface.

In outlining this system so far, nothing has been mentioned in regard to the mining of the ore which existed above the main or mining level and the paper has been written as though this ore had been exhausted, in order to avoid confusion and complications. As a matter of fact a considerable tonnage of ore in place existed above this level over some of the stopes and pillars. This ore was mined in two ways, (1) by "benching" and

(2) by carrying the "Latouche system" stopes through to the "bluff level," something over 90 ft. above the main or mining level. The latter was done by continuing the mining raises (*I*) the additional distance and driving suitable approaches on the bluff level. This made the entire lift of some of the stopes almost 250 ft., but access was had to the mining raises on the two levels, the main and the bluff.

Figs. 15, 16 and 17 are vertical sections through the orebody normal to the strike of the ore. These show the extent of the original ore deposit

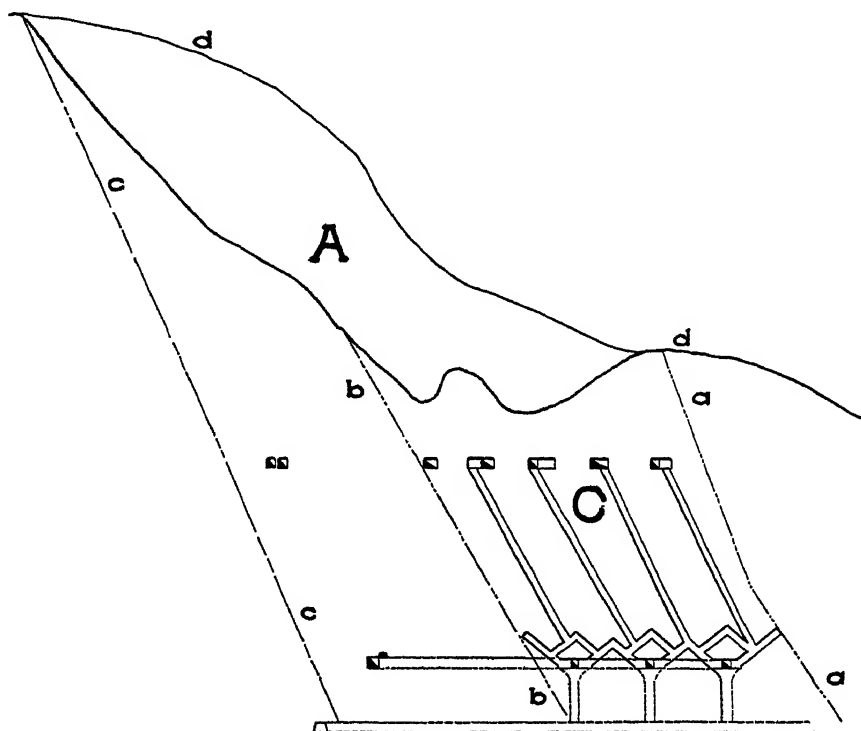


FIG. 17.—VERTICAL SECTION, NORMAL TO STRIKE, THROUGH NORTHERN PART OF OREBODY; SHOWING RELATIVE AREAS MINED BY VARIOUS METHODS.

A. Bluff mining.
C. Latouche system.
a. Hanging wall.

b. Economic footwall.
c. Limit of mineralization.
d. Original surface.

and the relative areas mined by bluff mining, benching and by the Latouche system.

The "benching" work was carried on in the hanging wall portions of several of the stopes. As soon as the mining raises in these areas were brought through, a grizzly was installed on the main level and a raise to the surface driven above the grizzly. These raises were then enlarged with Bull Moose rounds and the adjacent ore broken into the pits thus formed much the same as in glory-hole mining.

The cost of breaking ore by "benching" is given in Table 4 below, and in greater detail in Appendix B. Technical data covering "benching" are also given in Appendix G.

TABLE 4.—*Cost of Breaking Ground—Benching*
[Tons Broken, 119,868]

	AMOUNT	PER TON
Miners and helpers.....	\$10,370.43	\$0.087
Timber supplies.....	48.00	
Explosives.....	5,846.61	0.049
Machine drill expense.....	3,865.86	0.032
Miscellaneous and general mine expense.....	4,277.21	0.036
Total.....	\$24,408.11	\$0.204

Ore through Grizzlies—Bulldozing

The ore drawn from the stopes, pillars or "benching" pits is all put through the same type of grizzly, constructed of 100-lb. rails with 11-in. spaces between. A detail of the grizzly used is shown in Fig. 7. The large rocks are either broken with a hammer or block-holed as no "plastering" (or "bulldozing") is allowed. The term "bulldozing," however, is used to designate the operation of putting the ore through the grizzlies. Table 5 gives the cost of this operation in drawing ore from the three types of mining. This is given in greater detail in Appendix C.

TABLE 5.—*Cost of Ore through Grizzlies (Bulldozing)*
[Tons Bulldozed, 1,682,079]

	Stopes		Pillars		Benching		Total Average	
	Amount	Per Ton	Amount	Per Ton	Amount	Per Ton	Amount	Per Ton
Tons bulldozed.....	1,159,693		402,518		119,868		1,682,079	
Bulldozers.....	\$ 98,960.74	\$0.086	\$29,494.43	\$0.073	\$11,157.11	\$0.093		
Explosives.....	46,406.33	.040	13,880.86	.035	6,601.88	.055		
Machine drill expense	32,966.53	.028	11,375.06	.029	3,288.48	.027		
Thawing and flushing					521.06	.004		
Repairing chutes and grizzlies—labor....	9,988.00	.008	2,984.78	.007	265.55	.002		
Repairing chutes and grizzlies—supplies	6,996.73	.006	2,597.52	.006	311.31	.003		
Miscellaneous and general mine expense.....	49,715.56	.043	18,175.75	.045	4,130.74	.035		
Total.....	\$245,033.89	\$0.211	\$78,508.40	\$0.195	\$26,276.13	\$0.219	\$349,818.42	\$0.208

Table 6 gives the tons of ore drawn per man shift and the powder consumption per ton of ore. This is given in greater detail in Appendix H.

TABLE 6.—*Ore through Grizzlies; Showing Tons Drawn per Bulldozer Shift and Powder Consumption*

	Stopes	Pillars	Benching	Total and Average
Tons bulldozed.....	1,159,693.00	402,518.00	119,868.00	1,682,079.00
Tons per bulldozing shift..	63.00	74.00	59.00	65.00
Pounds powder per ton ore	0.16	0.15	0.25	0.16

Loading, Tramming and Hoisting

The total average cost for loading, tramming and hoisting for 1,682,079 tons was \$0.137 per ton of ore. This cost includes indirect and general mine charges. The hoisting cost averaged \$0.051 per ton and loading and tramming \$0.086 per ton. This is given in greater detail in Appendix D for each block of ground and for the ore derived from each type of mining.

Summary of Costs

Table 7 shows the average cost of all ore produced to be \$0.793. Appendix E gives in detail a recapitulation of costs of breaking ground, bulldozing, loading, tramming and hoisting for each block of ground and by each method of mining. From this it will be noted that the average cost of these operations in the stopes was \$0.510 per ton; in benching \$0.560 per ton and \$0.468 per ton for the pillars with an average of \$0.503 per ton covering the whole tonnage derived.

Appendix F is a recapitulation in detail showing the entire cost of production, including development and stope preparation, for each block of ground and by each method of mining. This shows a total cost of all ore derived to be \$0.793 per ton made up of the \$0.503 for breaking ground, bulldozing, loading, tramming and hoisting and the \$0.290 for development, stope preparation, etc., shown in Table 1.

Cost of Supplies and Wage Rates

In order that all the foregoing costs may be used for comparative purposes, there is shown in Table 8 cost of various supplies for the years 1922, 1923, 1924, 1925 and 1926, and in Table 9 is given the wage schedule covering the same years.

TABLE 7.—*Summary of Costs for Entire Orebody*

ORE PRODUCTION			
Breaking Ground Tons Broken	Place	Cost per Ton	Average Cost per Ton
1,618,212	Stopes	\$0.161	
119,868	Benching	0.204	
501,741	Pillars	0.137	
<u>2,239,821</u>			<u>\$0.158</u>
ORE THROUGH GRIZZLIES			
Tons Bulldozed	Place	Cost per Ton	Average Cost per Ton
1,159,693	Stopes	\$0.211	
119,868	Benching	0.219	
402,518	Pillars	0.195	
<u>1,682,079</u>			<u>\$0.208</u>
LOADING, TRAMMING AND HOISTING			
Tons Trammed	Place	Cost per Ton	Average Cost per Ton
1,159,693	Stopes	\$0.138	
119,868	Benching	0.137	
402,518	Pillars	0.136	
<u>1,682,079</u>			<u>\$0.137</u>
Total Ore Production.....			\$0.503
DEVELOPMENT, STOPE PREPARATION			
		Cost per Ton	Average Cost per Ton
Stope preparation.....		\$0.146	
Pillar preparation.....		0.042	
Prospect raises and cuts.....		0.004	
Ventilation raises and cuts.....		0.003	
Stope development.....		0.046	
Preliminary development.....		0.040	
Diamond drilling.....		0.009	
Total.....			\$0.290
Total average cost of ore derived.....			\$0.793

TABLE 8.—*Cost of Various Mine Supplies Laid Down at Latouche*

Material	Cost				
	1922	1923	1924	1925	1926
Grizzly rails, 100 lb. 12 ft. long each	\$12.91	\$16.18	\$14.86	\$13.34	\$12.35
Track rails, 35 lb., per 100 lb.....	2.83	3.74	3.89	2.72	2.52
Powder 40 per cent. Red Cross, per 50-lb. box.....	7.28	6.85	7.02	7.05	7.03
Powder 40 per cent. gelatin, per 50-lb. box.....	7.78	8.08	7.40	7.21	7.36
Caps No. 6, per 100.....	1.12	1.40	1.07	1.07	1.07
Caps No. 8, per 100.....	1.86	1.80	1.83	1.77	1.73
Caps No. 8, electric.....		0.178	0.185	0.173	0.182
Fuse, triple tape, per 100 ft.....	0.81	0.76	0.77	0.77	0.77
Fuse, Dreadnaught, per 100 ft....	0.59	0.65	0.63	0.63	0.65
Fir lumber, imported, per M ft. board measure.....	42.50	44.00	39.25	46.04	40.50
Spruce lumber, local cut, per M ft. board measure.....	32.26	28.50	26.85	22.47	27.33
Carbide, per 100 lb.....	7.12	6.68	6.77	6.77	6.77
Candles, per box.....	6.86	7.25	6.94	7.00	3.50
Blacksmith coal, per 100-lb. sacks..	2.78	1.35	1.45	1.86	1.10
Mild steel, per lb.....	0.036	0.041	0.040	0.039	0.039
Drill steel, 1½ in. hollow hexagonal, per lb.....	0.127	0.148	0.130	0.085	0.085
Drill steel, 1¼ in. hollow round, per lb.....	0.178	0.171	0.171	0.104	0.104
Drill steel, 1 in. quarter octagon (or cruciform), per lb.....	0.147	0.116	0.116	0.063	0.093

Remarks

This scheme, like most departures from established practice, is the adoption and development of the best ideas of the entire organization and cannot be credited to any one person. Changes have been made from the time of its inception to overcome difficulties encountered.

In applying this scheme of mining to the ore on the lower level, now being developed, several changes are to be made, the most important one being to change the width of stope from 70 to 60 ft. and increase the width of pillar from 30 to 40 ft. The distance of the mining raise from the pillar will be reduced from 15 to 13½ ft. The 30-ft. pillar was found to be too narrow in the longer stopes and would not stand up until mining could be carried to the point desired. The narrow pillars became shattered and broken after undercutting had been completed or after the first round blasted and made mining in them somewhat hazardous.

TABLE 9.—*Wage Schedule at Latouche, Showing Rates Paid per 8-hr. Shift*

	1922	1923	1924 to 1926
Miners.....	\$4.50	\$5.40	\$4.90
Miners' helpers.....	4.00	4.75	4.25
Shovelers.....	4.00	4.75	4.25
Grizzly men.....	4.50	5.40	4.50
Ore pocket grizzly helpers.....	4.25	5.00	4.50
Cagers.....	4.25	5.00	4.50
Skip tenders.....	4.25	5.00	4.50
Hoistmen.....	4.50	5.40	4.90
Motormen.....	4.25	5.00	4.50
Steel sharpeners.....	4.75	5.65	5.15
Steel sharpeners' helpers.....	4.00	4.75	4.25
Timber framer.....	4.50	5.40	4.90
Mechanics.....	4.75	5.65	5.15
Mechanics' helpers.....	3.75	4.50	4.00
Head tracklayer.....	4.50	5.40	4.90
Tracklayers.....	4.25	5.15	4.65
Tracklayer, helpers.....	4.00	4.75	4.25
Track cleaners.....	4.00	4.75	4.25
Timbermen.....	4.50	5.40	4.90
Timbermen helpers.....	4.00	4.75	4.25
Loaders.....	4.25	5.00	4.50
Tool nippers.....	4.00	4.75	4.25
Pipemen.....	4.50	5.40	4.90
Pipemen helpers.....	4.00	4.75	4.25
Trammers.....	4.00	4.75	4.25
Outside laborers.....	3.65	4.25	3.90
Powder and fuse men.....	4.25	5.15	4.65

Advantages of Method

1. *Safety*.—There has been but one fatal accident during the time of mining the entire tonnage. This occurred when a crew of miners failed to bar down the back of the bench after springing one of the long rounds. A piece of rock fell from the back striking one of the miners who was loading the round.

2. Low powder consumption.

3. Large tonnage per machine drill shift or labor shift employed.

4. Low costs (as a result of 2 and 3).

5. Ability to mine large tonnage rapidly after preparation work finished.

Disadvantages of Method

1. The large expenditure for grizzly chambers and other preliminary work would preclude using this method unless there is a fairly large tonnage of ore above the grizzly level. A level must be available for

bulldozing; whether grizzlies are used or not depends on the size of the cars and chutes.

2. In common with most methods for breaking large tonnages, it is not possible to leave any low-grade ore occurring between the undercut and the level above.

3. Comparatively lengthy period of time to get a block of ground producing due to extensive preliminary work.

ACKNOWLEDGMENTS

The writer wishes to express his appreciation of the services rendered by C. B. Hull⁵ and C. W. Poy⁶ in the preparation of the drawings accompanying this article; also suggestions made by B. B. Nieding.⁷

⁵ Chief Engineer, Latouche plant.

⁶ Assistant Engineer, Latouche plant.

⁷ Manager, Alaska Mines of Kennecott Copper Corpn.

APPENDIX A.—STOPE PREPARATION, JAN. 1, 1922, TO DEC. 31, 1926.—(CONTINUED)

	205 L.A.	206 L.A.	205 H.E.	206 P.E.	Total	Per Foot	Total	Per Ton
Advances	Amount	Amount	Amount	Amount	Amount	Amount	Amount	Amount
STOPE PREPARATION								
Chutes	1	1005.04	1	631.48	1	1119.19	1	679.74
Ordinary Cut Out	100	842.81	100	853.91	100	682.04	100	662.57
Grizzly Installation	2	1523.06	2	353.78	2	1474.47	2	1474.47
Grizzly Level	165	2192.97	325	5935.95	160	2955.78	310	3437.74
Raisins to Stope	48	331.64	47	289.43	320	636.40	70	491.35
Mining Raisins	615	5960.90	685	5670.30	402	3393.03	42	803.84
Undercutting Raisins	317	2434.48	286	1465.84	213	2337.39		
Undercutting (sq. ft.)	6898	4378.46	6850	3475.87	4440	3739.04		
Drifts & Crosscuts	137	985.56	69	1015.97	116	1697.35	8	149.04
Timbering		67.70		272.88				
Total.....		19510.07		18716.44		18204.69		6314.35
PIVAL PREPARATION								
Chutes								
Grizzly Raisins					791			4499.88
Grizzly Cut Out					25			6632.85
Grizzly Installation					270			132.48
Raisins to Stope	40	287.92					15	3754.32
Filler Mining Raisins	120	947.99			260			6622.86
Undercutting Raisins	85	611.84			2694			28447.78
Undercutting (sq. ft.)	400	409.94			2089			13187.60
Drifts & Cross cuts	86	517.25	35	335.47	28971			37295.70
Timbering					5060			22589.00
Prospect Raisins								3310.15
Total.....		1774.94		635.67	47			440.65
STOPE DEVELOPMENT								
South Drift	45	2064.61	96	2806.20	65	2065.61	96	2806.20
South Lift	186	4122.67	166	4546.60	223	4171.71	129	2557.15
S.H.W. Drift	35	789.17	45	924.09	20	452.64	27	549.76
W.C. Drifts, W.C. & Raisins								
Total.....		6916.09		8776.99		6544.16		7122.18
PRELIMINARY DEVELOPMENT								
Sump & Deslimes	1276.27		1759.72		637.86		886.51	
New Slaking Hoist	174.38		235.32		72.36		118.87	
Ore Pocket	775.23		1087.64		335.44		532.48	
Main Battery	675.64		935.01		250.87		497.62	
Aerial Span	69.60		55.01		50.72		46.40	
201 Manway	264.20		345.68		179.08		179.08	
Total.....	5801.48		4467.65		1306.41		2759.52	
DIAMOND DRILLING	597.68		946.02		276.90		476.16	
GRAND TOTAL.....		2801.06		2516.22		24699.14		20921.32

APPENDIX B.—Cost for Breaking Ground, Jan. 1, 1922, to Dec. 31, 1926

CA	800	Per	801	Per	802	203	Per	204	Per	205	Per	206	Per	GRAND TOTAL					
Amount	Subst	Amount	Per	Amount	Subst	Amount	Per	Amount	Subst	Amount	Per	Amount	Subst	Per					
21142	148117	388303	524430	287882	70031	309408	143083	27900	2324621										
Tons Broken																			
12202	94896	238219	302070	132579	213842	258585	143083	27900	1018219										
Miners & Helpers																			
819,80	0,67	5234,35	0,67	9564,50	0,40	14542,14	0,48	9562,33	0,52	12529,58	0,59	4207,25	0,50	14037,88	0,53	1482,20	0,53	671,30	0,54
Timber Supplies																			
251,63	0,03	885,37	0,04	908,21	0,03	4,16	-	204,36	0,01	247,87	0,01	1,17	-	835,75	0,06	1,37	-	48,00	0,01
Explosives																			
527,19	0,77	4286,64	0,44	7050,98	0,22	1790,79	0,02	7952,30	0,39	8235,17	0,38	3356,98	0,48	9046,14	0,54	3864,44	0,59	8604,77	0,61
M.D. Expense																			
491,64	0,40	4071,45	0,40	6893,10	0,33	4379,76	0,28	6893,99	0,31	7650,18	0,31	2699,97	0,39	6894,76	0,34	3384,44	0,56	5282,17	0,65
Misc. O.M. Expense																			
301,22	0,51	4286,62	0,46	4149,48	0,22	7664,97	0,28	3966,06	0,21	5799,34	0,28	1821,72	0,26	5790,13	0,22	1980,66	0,33	5134,82	0,44
Total.....																			
2709,15	2,23	18539,38	1,95	31612,43	1,32	27659,04	1,15	34509,74	1,62	12085,92	1,12	38352,80	1,43	13401,79	1,22	34036,66	1,29	9116,64	1,16
BREAKING																			
8940	35669	52023	13356															119668	
Miners & Helpers																			
770,80	0,86	3522,13	0,89	4318,13	0,53	1756,65	0,21											1837,43	0,57
Timber Supplies																			
46,98	0,01	1,02	-															48,00	0,01
Explosives																			
628,36	0,71	1542,81	0,39	2703,81	0,52	971,54	0,50											5610,11	0,69
M.D. Expense																			
167,04	0,62	1027,61	0,26	1884,91	0,36	486,59	0,28											3687,96	0,52
Misc. & O.M. Exp.																			
331,78	0,34	1317,60	0,39	1952,47	0,35	552,76	0,30											4577,11	0,54
Total.....																			
1,659,37	2,43	7592,45	1,13	10700,34	1,207	3762,60	1,15											2406,11	1,14
211428																			
Tons Broken																			
64211	104815	170346	35644	74040	8994	42582												501741	
Miners & Helpers																			
4046,97	0,75	3986,60	0,36	5618,35	0,24	2484,00	0,44	2325,55	0,31	343,76	0,58	3055,44	0,54					17282,77	0,48
Timber Supplies																			
56,44	-	322,89	0,02	322,89	0,02	181,64	0,04	140,72	0,02	24,53	0,03	250,95	0,05					11341,43	0,32
Explosives																			
2539,12	0,48	2741,24	0,22	2884,50	0,22	1650,60	0,47	977,37	0,18	104,31	0,12	2843,98	0,62					14682,12	0,31
M.D. Expense																			
2781,46	0,81	1780,24	0,16	4437,03	0,25	1623,40	0,44	1404,79	0,20	241,54	0,27	2790,83	0,61					13169,04	0,30
Misc. O.M. Exp.																			
1260,67	0,58	1108,18	0,13	1729,61	0,16	761,62	0,17	1801,63	0,21	935,77	0,52	2767,05	0,49					3264,64	0,29
Total.....																			
1260,67	0,58	1108,18	0,13	1729,61	0,16	761,62	0,17	1801,63	0,21	935,77	0,52	2767,05	0,49					3264,64	0,29
GRAND TOTAL																			
4331,63	2,24	30748,46	2,08	50645,03	1,30	72471,99	1,38	39595,92	1,55	40939,81	1,48	18798,77	1,64	52059,66	1,92	34076,65	1,94	9116,64	1,16
BREAKING 030203																			
																		4830,44	1,58

APPENDIX C.—COST OF ORE THROUGH GRIZZLIES, JAN. 1, 1922, TO DEC. 31, 1926

	204		205		206		207		208		209		210		211		212		213		214		215		216		217		218		219		220		221		222		223		224		225		226		227		228		229		230		231		232		233		234		235		236		237		238		239		240		241		242		243		244		245		246		247		248		249		250		251		252		253		254		255		256		257		258		259		260		261		262		263		264		265		266		267		268		269		270		271		272		273		274		275		276		277		278		279		280		281		282		283		284		285		286		287		288		289		290		291		292		293		294		295		296		297		298		299		300		301		302		303		304		305		306		307		308		309		310		311		312		313		314		315		316		317		318		319		320		321		322		323		324		325		326		327		328		329		330		331		332		333		334		335		336		337		338		339		340		341		342		343		344		345		346		347		348		349		350		351		352		353		354		355		356		357		358		359		360		361		362		363		364		365		366		367		368		369		370		371		372		373		374		375		376		377		378		379		380		381		382		383		384		385		386		387		388		389		390		391		392		393		394		395		396		397		398		399		400		401		402		403		404		405		406		407		408		409		410		411		412		413		414		415		416		417		418		419		420		421		422		423		424		425		426		427		428		429		430		431		432		433		434		435		436		437		438		439		440		441		442		443		444		445		446		447		448		449		450		451		452		453		454		455		456		457		458		459		460		461		462		463		464		465		466		467		468		469		470		471		472		473		474		475		476		477		478		479		480		481		482		483		484		485		486		487		488		489		490		491		492		493		494		495		496		497		498		499		500		501		502		503		504		505		506		507		508		509		510		511		512		513		514		515		516		517		518		519		520		521		522		523		524		525		526		527		528		529		530		531		532		533		534		535		536		537		538		539		540		541		542		543		544		545		546		547		548		549		550		551		552		553		554		555		556		557		558		559		560		561		562		563		564		565		566		567		568		569		570		571		572		573		574		575		576		577		578		579		580		581		582		583		584		585		586		587		588		589		590		591		592		593		594		595		596		597		598		599		600		601		602		603		604		605		606		607		608		609		610		611		612		613		614		615		616		617		618		619		620		621		622		623		624		625		626		627		628		629		630		631		632		633		634		635		636		637		638		639		640		641		642		643		644		645		646		647		648		649		650		651		652		653		654		655		656		657		658		659		660		661		662		663		664		665		666		667		668		669		670		671		672	
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APPENDIX F.—RECAPITULATION OF TOTAL COST FOR STOPES, JAN. 1, 1922, TO DEC. 31, 1926

	1920			201			202			203 E. J.			203 P. M.		
	Amount	Ton	Per Ton	Amount	Ton	Per Ton	Amount	Ton	Per Ton	Amount	Ton	Per Ton	Amount	Ton	Per Ton
Total Tons Est.	23189	74200	182700	381200	536000	294800	341400								
Tons Mined in Development	2347	310	12423	2703	2975	14490	9606								
Tons Broken Stoping	21142		149117	300303	82438	250599	287382								
Tons Bulldozed-Load-Track-Hoist	21142		25019	252496	457366	218016	291466								
Total Ore Breaking	\$12466.08	.590	40495.82	.898	147090.48	.460	274041.32	.470	153546.31	.542	139335.74	.460			
Stope Preparation	4761.66	.205	1830.84	.021	32026.66	.180	51057.93	.134	70585.91	.132	45460.71	.164	34015.03	.102	
Pillar Preparation			522.28	.013	17083.46	.093	16551.22	.044	24633.23	.065	2106.93	.021	12704.48	.037	
Prospect Hauls & Shuts	19.06	.001	182.61	.001	327.03	.001			309.65	.001	547.99	.001			
Ventilation Hauls									756.27	.002			2695.53	.038	
Ventilation Crosscuts									147.32	-	225.11	-	2545.84	.008	
Stope Development	7.76	-	117.15	.002	6707.19	.033	17666.64	.046	21461.95	.041	12084.37	.044	12133.01	.036	
Preliminary Development	738.66	.032	638.81	.008	7482.17	.041	15674.87	.041	77165.18	.041	17141.54	.041	14015.06	.041	
Diamond Drilling	202.61	.009	1650.48	.009	3339.34	.009	4775.72	.009	7531.93	.009	3033.16	.009			
TOTAL COST.....	\$ 16196.77	.837	3249.36	.344	106668.38	.955	221907.86	.735	373858.56	.760	235078.33	.820	219277.96	.714	

APPENDIX F.—RECAPITULATION OF TOTAL COST FOR STOPES, JAN. 1, 1922, TO DEC. 31, 1926
(CONTINUED)

	1924 M.V.			1925 F.V.			1926 F.V.			1927 F.V.			GRAND TOTAL		
	Per Amount Ton	Per Amount Ton	Per Amount Ton	Per Amount Ton	Per Amount Ton	Per Amount Ton	Per Amount Ton	Per Amount Ton	Per Amount Ton	Per Amount Ton	Per Amount Ton	Per Amount Ton	Per Amount Ton	Per Amount Ton	Per Amount Ton
Total Tons Est.	118900	874400	79700	109000	32470	58900	265259								
Tons Mined in Development	14044	15535	7200	5764	4871	1487	156897								
Tons Broken Stopping	79031	809508	60356	145083	27800		2259821								
Tons Bulldozed-Load-Team-Toilet	63572	198220	63989	21778	9027		1688079								
Total Ore Breaking	35156.97	513	120920.00	520	39974.56	605	80882.64	564	7782.32	478			934884.74	508	
Stope Preparation	31455.92	264	48820.97	122	19510.07	244	18718.84	172	128504.69	561	6214.24	117	881501.19	146	
Pillar Preparation	13795.85	116	9196.80	024	2774.94	035	395.67	004					109104.85	042	
Prospect Entries & Exits	396.71	003	315.82	001	2087.87	026	91.14	001	1110.59	024	6145.19	115	11473.71	004	
Ventilation Entries									1082.54	033			4654.31	002	
Ventilation Crosscuts													2649.59	001	
Stope Development	6354.97	071	12830.25	034	5918.85	087	8376.99	076	6852.16	205	5912.18	112	119231.49	046	
Preliminary Development	4881.05	041	15472.62	041	3231.42	041	4467.56	041	1346.63	041	2178.53	041	174457.57	040	
Diamond Drilling	1089.07	009	5354.35	009	597.88	009	569.02	002	232.58	008	472.76	002	25512.21	009	
TOTAL COST..... \$	95564.64	1.017	207920.98	761	67175.37	1.045	82298.66	687	36481.46	1.561	20921.32	595	160104.46	793	

APPENDIX G.—BREAKING GROUND, JAN. 1, 1922, TO DEC. 31, 1926

SHOVELS												
Machine	Tons Broken	Drill Per Shift	Tons Per Shift	Labor Shift	Tons Per Shift	Pounds Powder Per Ton Ore	Number Rounds	Number Holes Drilled	Total Feet Drilled	Average No. Holes Per Round	Average Ft. Drilled Per Round	Tons Ore Broken Per Foot of Hole
04	12,202	74	165	284	55	5569	4	247	2405	62	601	5.07
200	52,896	415	225	715	131	22864	18	1633	21597	90	1200	4.25
201	289,819	792	808	1346	176	44114	44	3602	28013	87	750	7.25
202	302,070	1411	214	2083	145	60,326	66	4,281	53,828	64	816	5.61
203 H.W.	132,879	386	208	1,450	115	48,145	44	2,465	27,987	57	682	6.42
205 P.W.	213,842	1,064	197	1,865	113	54,236	44	2,780	35,350	57	749	5.94
204 H.W.	70,097	1,064	160	651	63	22,412	25	2,009	11,274	40	451	5.21
204 P.W.	243,526	1,270	205	1,949	135	57,424	55	3,411	45,464	64	886	5.80
205 H.W.	60,460	877	105	980	68	22,985	27	2,411	14,894	46	638	4.19
205 P.W.	143,003	915	125	1,444	79	50,497	34	2,914	39,844	61	1,092	5.67
206 H.W.	27,300	220	147	304	92	7,612	12	515	6,132	42	512	4.54
Total & Average	1,618,212	6,156	196	15,172	123	895,091	24	23,099	292,722	61	773	5.52
HEMISLES												
04	8,940	80	112	179	50	3,644		286	1,953			4.58
201	59,549	153	259	651	61	9,497		217	2,782			14.22
202	82,023	309	166	922	56	17,373		534	7,004			7.45
203 H.W.	19,234	63	233	354	55	6,133		115	1,052			12.46
Total & Average	119,866	645	192	2,106	57	36,507		1,155	13,893			9.02
PILARS												
200	84,221	281	195	280½	98	12,715	9	1,096	16,081	122	1,767	3.37
201	109,515	126	849	647	800	14,431	4	421	6,386	109	1,074	17.39
202	170,845	217	639	706	241	13,725	7	906	14,589	129	2,080	11.91
203 H.W.	28,644	142	372	456	85	9,280	25	441	6,213	108	1,029	6.22
203 P.W.	74,060	198	374	322	230	5,947	7	692	6,403	65	1,201	6.61
204 H.W.	8,994	29	321	64	140	970	1	86	936	66	936	9.61
204 P.W.	45,182	225	125	131	78	14,028	3	898	12,093	101	1,000	2.13
Total & Average	501,741	1,825½	376	324½	135	78,444	18	4,240	64,496	108	1,612	7.76
GRAND TOTAL & AVERAGE	2,229,821	10,110½	221	10,843½	121	809,442	23	419	270,721	66	885	6.04

APPENDIX H.—ORE THROUGH GRIZZLIES, JAN. 1, 1922, TO DEC. 31, 1926

SHIFT	Tons Bulldozed	Shifting Shifts	Tons per Shifting Shift	Pounds Fender	Pounds Grinding Per Ton Ore
NO. 1					
04	12,202	186	64	3,905	.38
200	17,084	236	72	2,705	.16
201	149,187	2027	71	14,063	.10
202	267,448	4010	61	24,207	.10
203	382,815	5592	61	38,592	.10
204	191,319	2924	65	45,574	.28
205	54,678	812	66	14,100	.26
206	170,776	2844	59	24,305	.14
207	53,989	942	57	14,766	.27
208	51,778	626	83	6,882	.12
209	5,087	121	75	2,176	.23
Total & Average.....	1,109,693	18274	63	186,622	.16
NO. 2					
04	8,940	154	64	3,844	.53
201	39,569	746	53	6,509	.21
202	65,083	884	62	13,575	.26
203	19,336	289	67	4,254	.22
Total & Average.....	119,668	2036	59	29,983	.25
NO. 3					
200	7,936	64	119	1,136	.14
201	103,759	1455	71	14,023	.14
202	137,890	1784	77	19,774	.14
203	16,864	261	55	3,331	.21
204	183,137	1821	77	14,525	.15
205	8,934	212	41	4,411	.19
206	27,924	395	71	4,361	.16
Total & Average.....	408,510	5468	74	59,571	.16
GRAND TOTAL & AVERAGES.....	1,688,079	25777	65	276,179	.16

DISCUSSION

R. R. VAN VALKENBURGH, Wharton, N. J.—Undercutting costs were kept on a square-foot basis. We took this up with the author and others familiar with it. There seems to be no particular reason except that the work was contracted and the undercut volume was of no particular interest. The square foot advance was more important.

G. A. PACKARD, Boston, Mass.—Is not the time of preparation very considerably more than in the ordinary shrinkage stope?

R. R. VAN VALKENBURGH.—Yes, but there are many points of attack in this system. It is reasonable, therefore, to expect that speed could be obtained if necessary.

G. A. PACKARD.—Could not the preparation be done as quickly as in an ordinary shrinkage stope?

R. R. VAN VALKENBURGH.—No, because there is so much more of it.

G. A. PACKARD.—It was tried on a block 280 by 800 ft. How small an orebody do you think could be treated satisfactorily in this way?

R. R. VAN VALKENBURGH.—I have always thought of adapting it in the other direction; namely, with the strike of the orebody instead of across it. In order to get the maximum result there should be a width of at least 40 ft., which is the distance between the working raises.

G. A. PACKARD.—It is obvious that all of the preparatory work could not be undertaken for an orebody of very limited thickness.

B. F. TILLSON, Franklin, N. J.—Could one apply this method to an orebody 60 or 80 ft. thick? What experience can we get in that respect?

G. A. PACKARD.—Would it be applicable to the sometimes comparatively thin beds of the Joplin district, for example?

R. R. VAN VALKENBURGH.—No; based on costs I know of in Joplin.

B. F. TILLSON.—Relative to other methods of mining?

R. R. VAN VALKENBURGH.—I do not think it would apply to relatively thin deposits of 60 feet.

F. W. SPERR, Houghton, Mich.—Is not this deposit supposed to go down hundreds of feet lower than shown in the illustrations?

G. A. PACKARD.—I believe it is.

F. W. SPERR.—The hanging wall must be caved in from the foot wall or ore will be lost.

G. A. PACKARD.—According to Fig. 13 the hanging-wall side is broken first; then the broken ore is allowed to pile up on that side.

F. M. RADEL, Wharton, N. J.—At Latouche the hanging wall dips at about 35°. It is a slate formation. Ore is piled up on the hanging wall as a support to the wall. This is accomplished by drawing the footwall chutes faster than those near the hanging and is made possible by chute spacing. That seems to be the most important feature.

No ore is lost, so far as I know. I worked there for five years and helped to start the Latouche system. I may be prejudiced, therefore, in thinking that no other system could have been better adapted. We started back-stoping with the ordinary shrinkage method. Tons per man broken have been increased 1000 per cent. since then.

The most important advantage is the safety. It is unnecessary to go into the stope. The ground had cracks and slips, was very soft and gave no warning; hence was treacherously dangerous. Since the system has been put in, there has been but one fatality and that attributable in no way to the mining method.

There are many other factors, of course, that recommend the Latouche system. Access to the stopes at many points eliminates all temporary pipe installations for air and water. Also, in these raises are provided small hoists for material and tools. Therefore care of drill steel and tools is made easy, and the work is made attractive to a better class of men.

B. F. TILLSON.—This system is of great interest to mining men. It is evidently a variant of the whole system in mining. It was probably 19 years ago that in Michigan, in the Iron River district, I found sublevel stoping in use. This practice was based on a development of drifting instead of a raising development, and the mine was satisfactorily operated in that way beneath the river. I regard it as a very valuable mining system. It has been pointed out that shrinkage stoping was employed in one of the mines in the Tri-State district, and I wonder why this system would not have made an increase over the shrinkage stoping tried at Beatson mine as opposed to the room-and-pillar method which was found advantageous.

There seems to be a feeling that the sublevel drift as the point of attack is in many local conditions superior to an attack upon a raise development. I wonder why the particular conditions under which these different methods of development are advantageous cannot be clarified. At Copper Hill, a system quite akin to the Latouche system was tried, and it was found desirable to vary it somewhat because of local conditions.

The condition of back, shape, length and thickness of a particular orebody undoubtedly has an influence as to the best way of development. A clarification of features would be very valuable.

F. M. RADEL.—It was thought that subleveling would be relatively more costly. In Arizona much of that is done. At stated distances a sublevel is driven and breaking is done from that. When the cost of driving the sublevel is compared to the cost of driving the raise, there is a considerable difference. That alone prevented us from going to the drift—we found that we could put up the raises at less cost.

G. A. PACKARD.—The character of the back undoubtedly is the deciding factor.

F. M. RADEL.—At Latouche the back is never worked under. Formerly, miners would go into the stope and bore for 2 or 3 hr. without directly producing. With this system there is no boring, except, as this paper indicates, in the first cut, which is so slight that it takes only $\frac{1}{2}$ hr. In other words, mining is continuous.

G. A. PACKARD.—How does the cost of preparation compare with the cost of preparation for the other methods?

F. M. RADEL.—The cost of preparation at Latouche is high, but it is more than offset by the cheap mining. The time it takes to break and how long one's money is tied up must be considered. In shrinkage stoping, the miners have to stand on the ore and there is a long time before that ore is drawn, but with the Latouche system it can be drawn as desired.

B. F. TILLSON.—Would it have been better to make it 250 ft. between levels instead of 200 feet?

F. M. RADEL.—It might have been. That is, of course, regulated by the dip.

A. NOTMAN, New York, N. Y.—I presume that the method would be applicable only to ground that stood pretty well. Although working in the solid one is not working on the solid. With the sublevel drifting method one is working on the solid.

G. A. PACKARD.—Would you consider, in these chambers that have been broken out, that a man was practically working on the solid? It is not exactly a parallel condition.

F. M. RADEL.—The system, I think, could be applied anywhere that sublevel stoping seemed applicable and even where the ground seems to have a greater tendency to cave.

A. NOTMAN.—The sublevel drifting scheme may be adapted to the ground. The question of development is not the only deciding factor between the two methods.

R. K. WARNER, New Haven, Conn.—It seems to me that the method is more or less limited to orebodies of a type that could normally be worked by ordinary shrinkage stoping. Dilution must not be a serious item. Under proper conditions the method becomes surprisingly flexible. One is able to adjust the work in the raises according as the ore breaks and need not worry about heavy patches of ore hurting men in the stope below. The work can be done there just as fast as the ore will break properly. In the academic study of mining methods the method is intermediate between the ordinary shrinkage stope and the block caved stope. By working in the raises, putting just enough pressure on the ore with explosives to make it come down right, orebodies too hard for ordinary caving or too soft for normal shrinkage can be handled economically. The method should be especially valuable where the strength of the back is likely to vary.

Mining Methods and Records at the United Eastern Mine

By ROY W. MOORE,* LOS ANGELES, CALIF.

(New York Meeting, February, 1928)

THE Tom Reed Extension and Big Jim mines of United Eastern Mining Co. are situated in the Oatman district, Mohave County, Ariz., about 28 miles southwest of Kingman, the nearest railway point.

Articles descriptive of the mine surface plant, mill construction, and plant operations have been written by Otto Wartenweiler,¹ the designing engineer; Wheeler O. North,² when he was assistant general manager for the company, and Earle M. Bagley, mill superintendent.³ The last article covers average costs for the full time the mill was operated.

The geology of the mines is described by F. L. Ransome⁴ of the United States Geological Survey.

Production was started in January, 1917, and the known orebodies were exhausted in May, 1925, showing a life of seven years and five months. On account of the comparatively short life and the fact that stoping operations were confined to two orebodies, little experimenting was done with mining methods.

The two orebodies were in what is generally called the Tom Reed-United Eastern vein, which occupies a fault fissure having a normal displacement of several hundred feet. Seven or eight very productive ore shoots were mined along this vein in a distance of $1\frac{1}{2}$ miles, the two worked by the United Eastern being 1 mile apart.

The maximum dimensions of the Tom Reed Extension ore shoot were: height, 750 ft.; length, 950 ft.; thickness, 48 ft. The vein structure varied from a solid quartz-calcite filling to a series of parallel stringers separated by andesite. From this body, 511,976 tons of ore were mined. (Fig. 1.)

The maximum dimensions of the Big Jim orebody were: height, 450 ft.; length, 850 ft., thickness, 35 ft. Its structure was similar to that of the Tom Reed Extension but larger sections of the vein were made up

* General Manager, United Eastern Mining Co.

¹ The United Eastern Mining and Milling Plant. *Trans.* (1918) **59**, 274.

² Mill Operations at United Eastern during 1917 and 1918. *Trans.* (1920) **63**, 548.

³ Operations at the United Eastern Mill. *Engng. & Min. Jnl.-Pr.* (1925) **119**, 436.

⁴ Geology of the Oatman Gold District, Arizona (A preliminary report). U. S. G. S. *Bull.* 743.

almost wholly of calcite and therefore were not of commercial value. From this orebody, 220,552 tons of ore were mined. (Fig. 2.)

SHAFTS, DRIFTS AND CROSSCUTS

The Tom Reed Extension shaft No. 1 was sunk for prospecting purposes. After a large tonnage of excellent ore had been proved on the

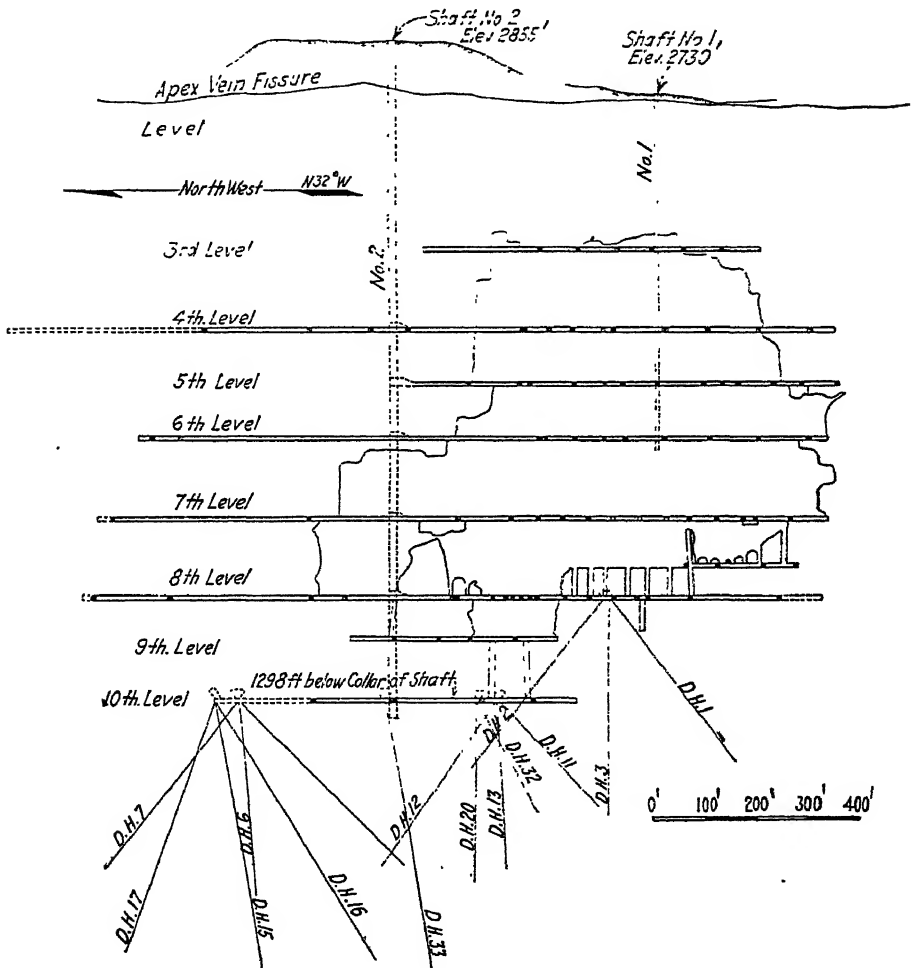


FIG. 1.—LONGITUDINAL SECTION THROUGH NO. 2 SHAFT STOPE, TOM REED EXTENSION.

third, fourth, fifth and sixth levels from this shaft, the main working shaft, known as No. 2, was sunk at a point 630 ft. to the north and about 450 ft. in the hanging wall from the apex of the vein fissure. It was located opposite the northerly end of the ore shoot, therefore subsequent settling of the hanging wall caused no inconvenience. (Fig. 3.)

This shaft was started at the top of a hill at an elevation 120 ft. above the collar of No. 1. The location afforded an excellent site for the mill in close proximity to the shaft. The shaft consists of three compartments, each 5 by 5 ft. inside dimensions. Two compartments were used for hoisting purposes and the other for ladderway, pump and air columns, and electric cables.

Ore pockets of 100 tons capacity were cut below each station on the fourth to the eighth levels inclusive. Each ore pocket discharged into two measuring pockets, each of one skip capacity. (Fig. 4.)

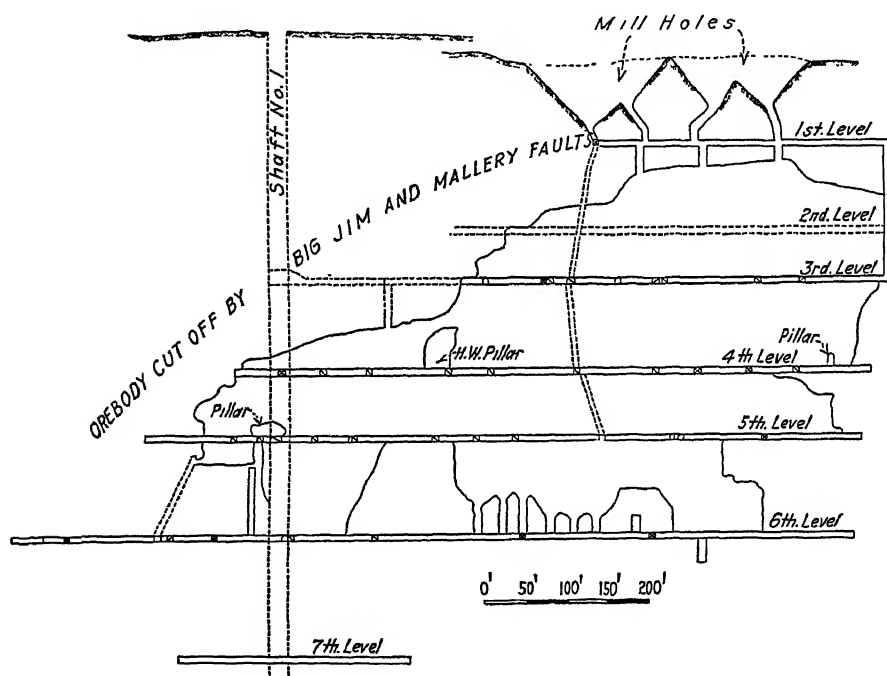


FIG. 2.—LONGITUDINAL SECTION THROUGH BIG JIM STOPE.

Levels were driven at intervals of 100 ft., 150 ft., and 200 ft. between the first level at 585 ft. and the lowest level at 1298 ft. The first level corresponds to the fourth level of shaft No. 1. The distances of the vein from the shaft on the top and bottom levels were respectively 60 and 260 feet.

The collar of the shaft was concreted to a depth of 50 ft. Timbers of merchantable grade were used throughout for shaft sets, the plates being 10 by 10 in. and the dividers 8 by 10 in. Stations were cut to the full length of the shaft, thus affording ample room on both sides of the double tracks leading to the station ore pockets. They were also large enough to permit storage of ore cars temporarily not in use, and were floored with planks 3 by 12 in. and $\frac{1}{4}$ -in. steel sheets, with the exception

of the grizzly sections, which were constructed of 35-lb. rails set 10 in. apart (Fig. 5). Crosscuts from the stations to the vein were double-tracked in part so as to permit passing of incoming and outgoing cars.

Development drifts in the vein proper measured 5 by 7.5 ft. and required no timbering, but sets were often required in prospect drifts in the vein fissure beyond the limits of the orebody. All levels were equipped with 18-in. gage track of 20-lb. rails.

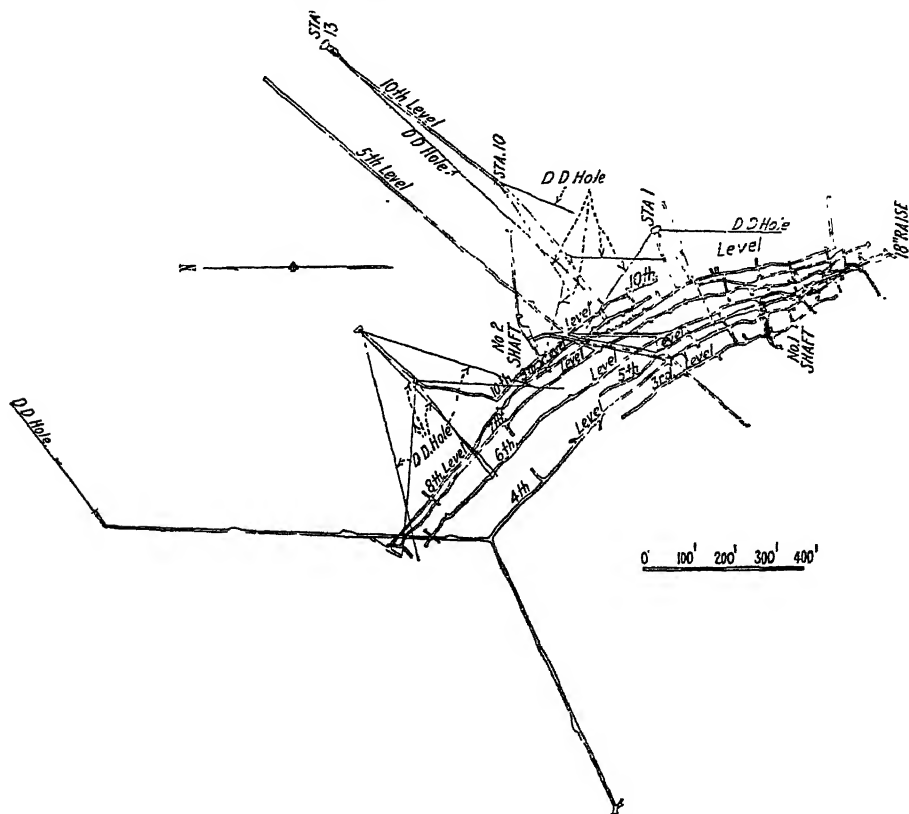


FIG. 3.—UNDERGROUND WORKINGS, UNITED EASTERN GROUP.

UNDERGROUND DEVELOPMENT PLANS

Each level was developed by drifts driven to the limits of the ore shoot and beyond. Raises were driven between levels as ventilation and ore-blocking requirements dictated. Forced ventilation through 10-in. galvanized pipe was used in headings, but the mine, as a whole, was ventilated by a suction fan having a capacity of 30,000 cu. ft., which was located at shaft No. 1.

Crosscuts were driven through the vein at 50-ft. intervals and at greater intervals were continued well into the walls to prospect for possible parallel veins.

Stopping

The horizontal cut and fill method was used almost exclusively in mining both orebodies. The untimbered rill was attempted, but was found not adaptable on account of the tendency of the vein to slab off. This tendency was due both to horizontal seams in the vein and to wall pressure.

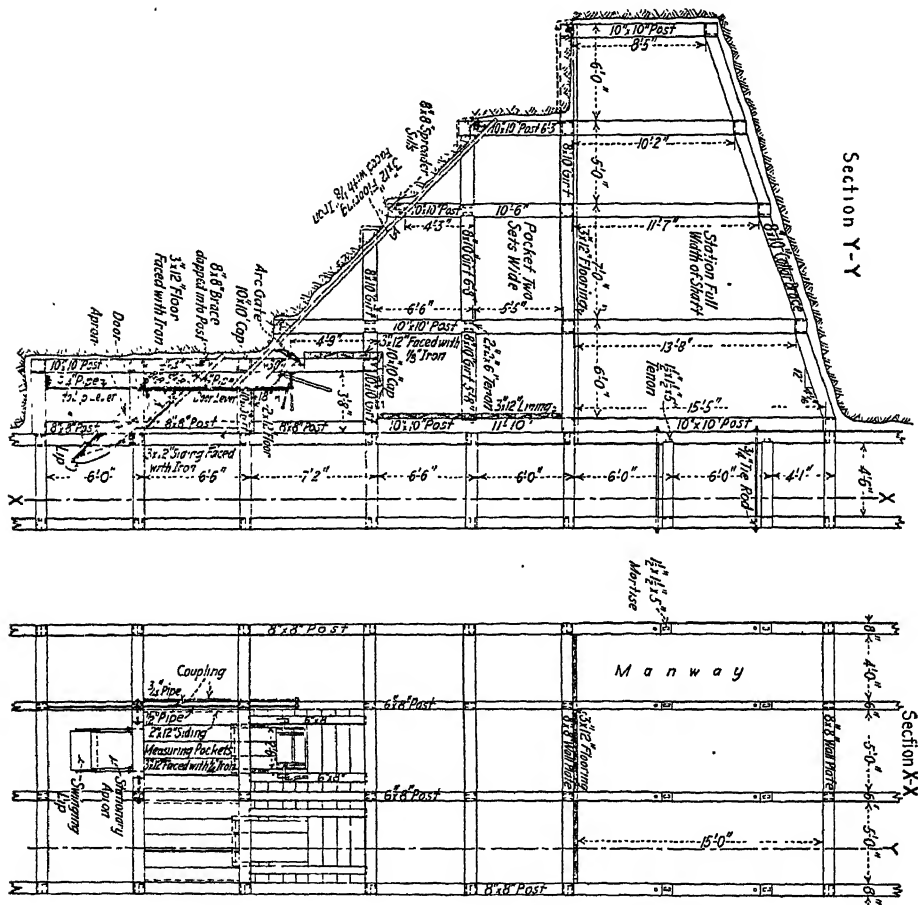


FIG. 4.—STATION AND POCKET, BIG JIM SHAFT.

The shrinkage method was used for small tonnages in narrow parts of the vein at the lower limits of both orebodies, but was not adaptable for the larger stopes for the reasons given above, and also on account of "sheeted" and soft hanging walls which would slough if not supported.

Use of the more costly method of stoping was justified as it resulted in a higher grade of ore being sent to the mill, and the treatment of a minimum tonnage over a minimum length of time necessary to exhaust the known ore supply, and therefore in a larger ultimate profit. The

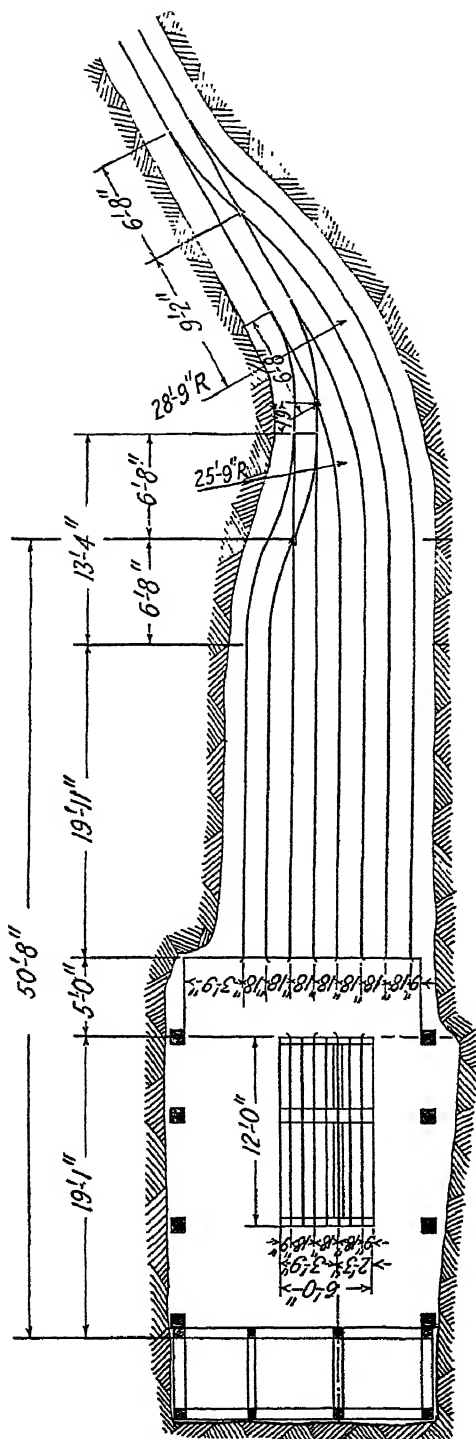


FIG. 5.—STATION PLAN, BIG JIM THIRD LEVEL.

method also afforded opportunities to prospect for possible near-by parallel veins by driving wall raises, the waste from which was used for stope filling.

In making preparations for stoping operations, the vein was cut out for its full width on the sill floor, beginning at the main crosscut from the shaft, and to a height to allow 6 ft. above the sill sets.

The first three levels were timbered two square sets high before filling was started, as it was believed that haulageways could be kept open to better advantage when the sets were subjected to great pressure, on account of the weight of fill and wall pressure. Later it proved inadvisable to use two sets as the timbers had a tendency to "jack-knife" under excessive weight. Thereafter, the timbers were placed only one set high across the vein. In both cases, the sets were topped with double thicknesses of 2-in. by 12-in. by 6-ft. cut lagging as fill support.

Cribbed manways and chutes constructed of 3 by 12-in. lumber were spaced at 22.5-ft. centers where the vein was not wider than 18 ft. Where the vein was wider, double rows of manway-chutes were constructed with the same longitudinal spacing. (Fig. 6.)

Drilling of stope backs to a depth of 7 ft. followed closely after completion of sill sets. The sets were then floored over, manway-chutes raised 3 ft. and filling placed to the same height. The fill was then properly leveled and floored over with 2-in. by 12-in. by 6-ft. cut lagging before the ore was blasted. After the ore was shoveled into chutes the floors were thoroughly swept and taken up in readiness for the usual 6 ft. of fill. Stopes were thus advanced by a succession of 6-ft. cuts of ore followed by corresponding benches of fill. Large pieces of ore were broken with doublejacks or by blockholing, so as to pass the 10-in. grizzlies over station ore pockets. (Fig. 5.)

Waste for filling was obtained from development and from stope-wall raises driven between chutes. These raises often extended to 60 ft. from the vein where the stopes were widest. They weakened the walls considerably and thus caused trouble when the stopes approached the levels above. At this latter stage the backs generally had to be supported by timber cribs. Waste was spread by use of small air hoists and scrapers in open stopes or by wheelbarrows in cramped quarters.

By the time it became necessary to draw ore from the Big Jim, that property had been pretty thoroughly developed. (Fig. 7.)

The same stoping method was used as in the Tom Reed Extension orebody, but operating conditions were much more favorable. The walls were more solid and consequently stood better. This often permitted two 6-ft. cuts of ore between periods of filling, thus eliminating one floor laying and taking up operation and a saving of time in chute building. A greater amount of the ore was drawn through the chute by gravity and thus shoveling expense was reduced.

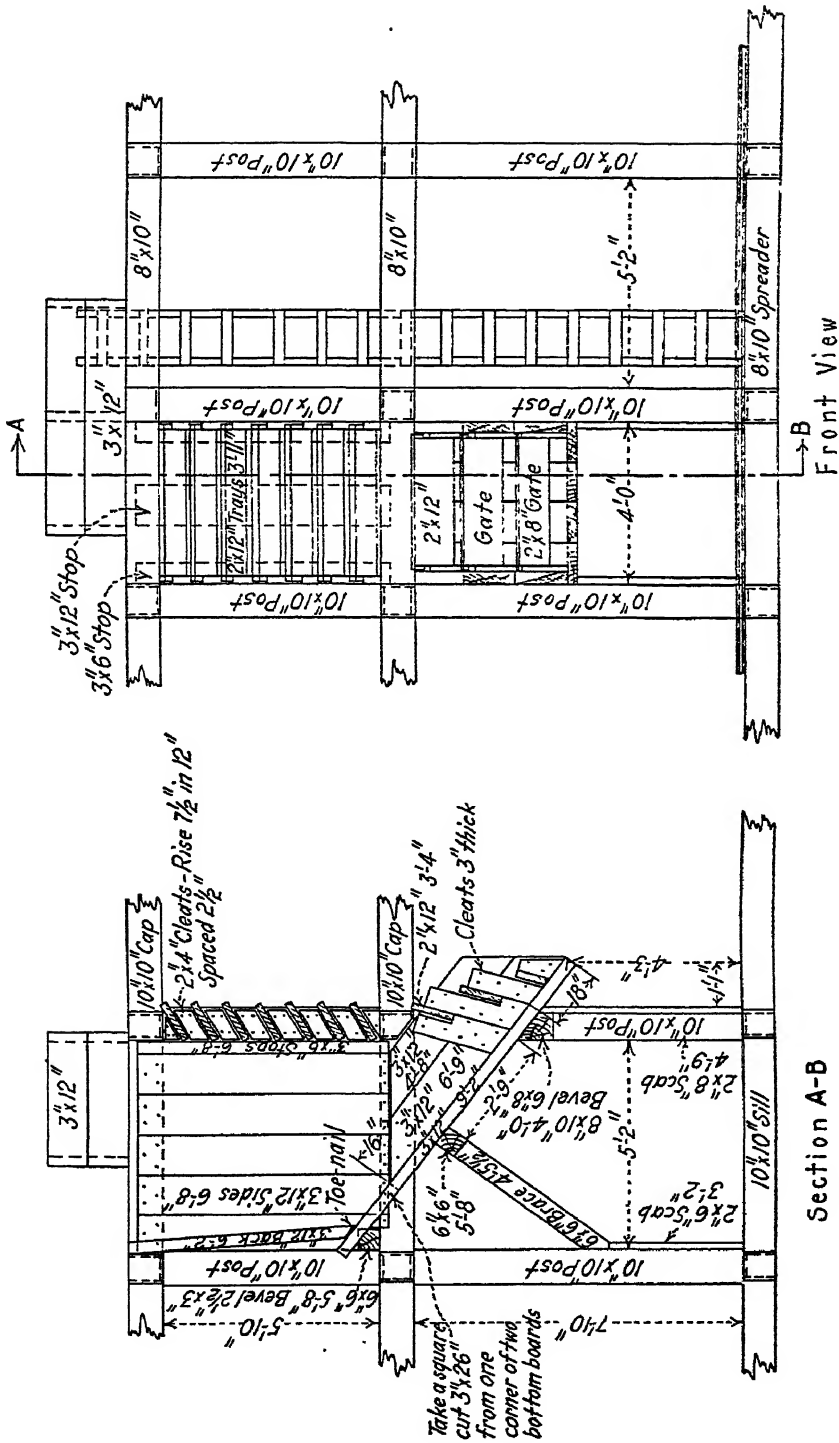


FIG. 6.—ORE CHUTE IN STANDARD SILL FLOOR TIMBERING.

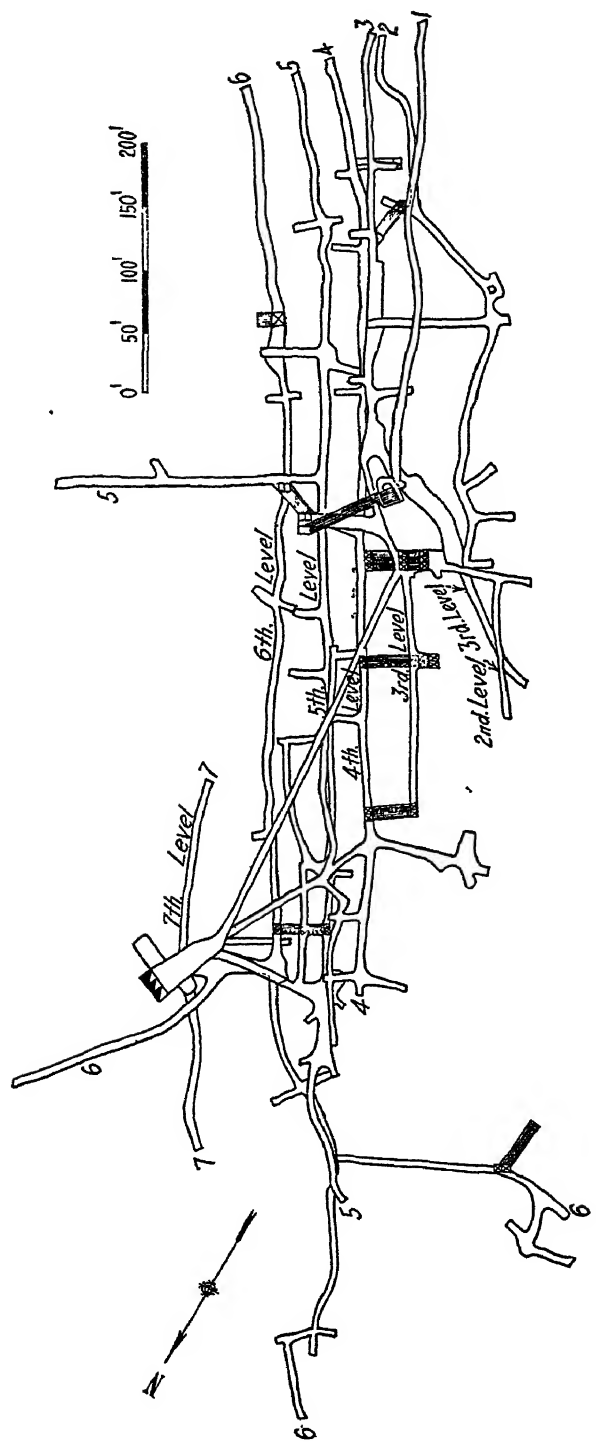


FIG. 7.—PLAN OF UNDERGROUND WORKINGS, BIG JIM MINE.

Waste was obtained more economically at the Big Jim. This ore-body apexed against the Mallery fault about 100 ft. below the surface. Raises were driven in ore from the third level to the Mallery fault and thence 15 to 20 ft. to the waste-distribution level, which was connected by raises to mill holes on the surface (Fig. 2). Waste could be dropped directly from the mill holes through three raises into the uppermost stope, or through a footwall raise, which connected one mill hole to the third, fourth and fifth levels, from which it was distributed to stope raises below those levels. This system resulted in a much lower filling cost than was obtained at the original property. Waste for filling was also derived from development work and from occasional stope-wall raises driven primarily to prospect for possible parallel veins.

Sill floors were cut out for the full width of the vein where the ore was of commercial grade, and sill sets only were placed. The sequence of mining operations was then the same as for the Tom Reed Extension orebody with the exception that two 6-ft. cuts of ore were often made in the stopes between periods of filling. Practically all of the filling was spread by use of scrapers and tugger or turbine air hoists, the use of which resulted in a material reduction of filling cost.

Power

Electric power was used for all purposes. Power was supplied by the Desert Power & Water Co. of Kingman, and transmitted at 44,000 volts to Oatman, where it was stepped down to 440 volts.

Hoisting Plants

The installation at the Tom Reed Extension shaft consisted of an Allis Chalmers double-drum electric hoist direct connected to a 150-hp. motor through a flexible coupling and herringbone gears.

Ore was hoisted in skips working in balance, each skip being suspended below a cage, which was used to hoist waste in cars as well as for men and supplies (Fig. 8). The skip loads averaged 2.2 tons each and the maximum hoisting speed was 800 ft. per minute.

A Wellman, Seaver, Morgan Co. hoist of like capacity was used at the Big Jim mine. The shaft equipment was practically a duplication of that used at shaft No. 2. Both headframes were of timber construction, 75 ft. in height, and were equipped with 7-ft. sheaves.

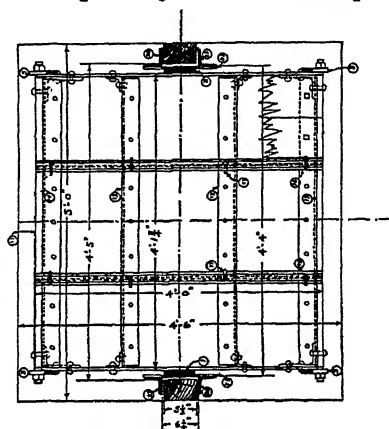
Compressor Plant

The installation at the Tom Reed Extension shaft consisted of two Ingersoll-Rand Imperial Type No. 10 belt-driven compressors 19 by 12 by 16 in., each furnishing 888 cu. ft. of free air per min. at 100-lb. pressure.

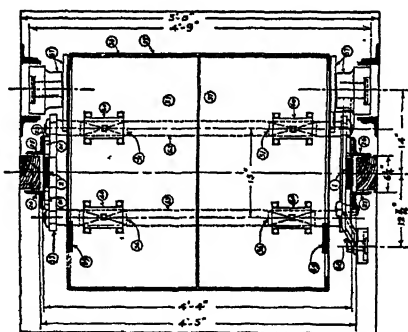
The Big Jim installation consisted of one machine of the above type and one machine of the same type 17 by 10 by 14 inches.

Pumping

The pumping problem was not serious at either mine. The water could be controlled at the Tom Reed Extension shaft by pumping 32 gal. per min. during 16 hr. per day. Water was pumped from the tenth



PLAN OF SECTION A-A



PLAN OF SKIP

FIG. 8.—ASSEMBLY OF CAGE AND 2-TON SKIP, BIG JIM MINE.

(1298-ft.) level to the eighth (1098-ft.) level and thence to the surface.

At the Big Jim, the flow could be handled ordinarily in 16 hr. by pumping 150 gal. per min. when working below the sixth level, above which the mine was dry. The pump columns at both mines were 4-in. standard black pipe.

Distribution of Compressed Air and Water

Air mains in both shafts were 4-in. standard black pipe; 3-in. mains were led off at each level and were reduced to 2-in. pipes as conditions

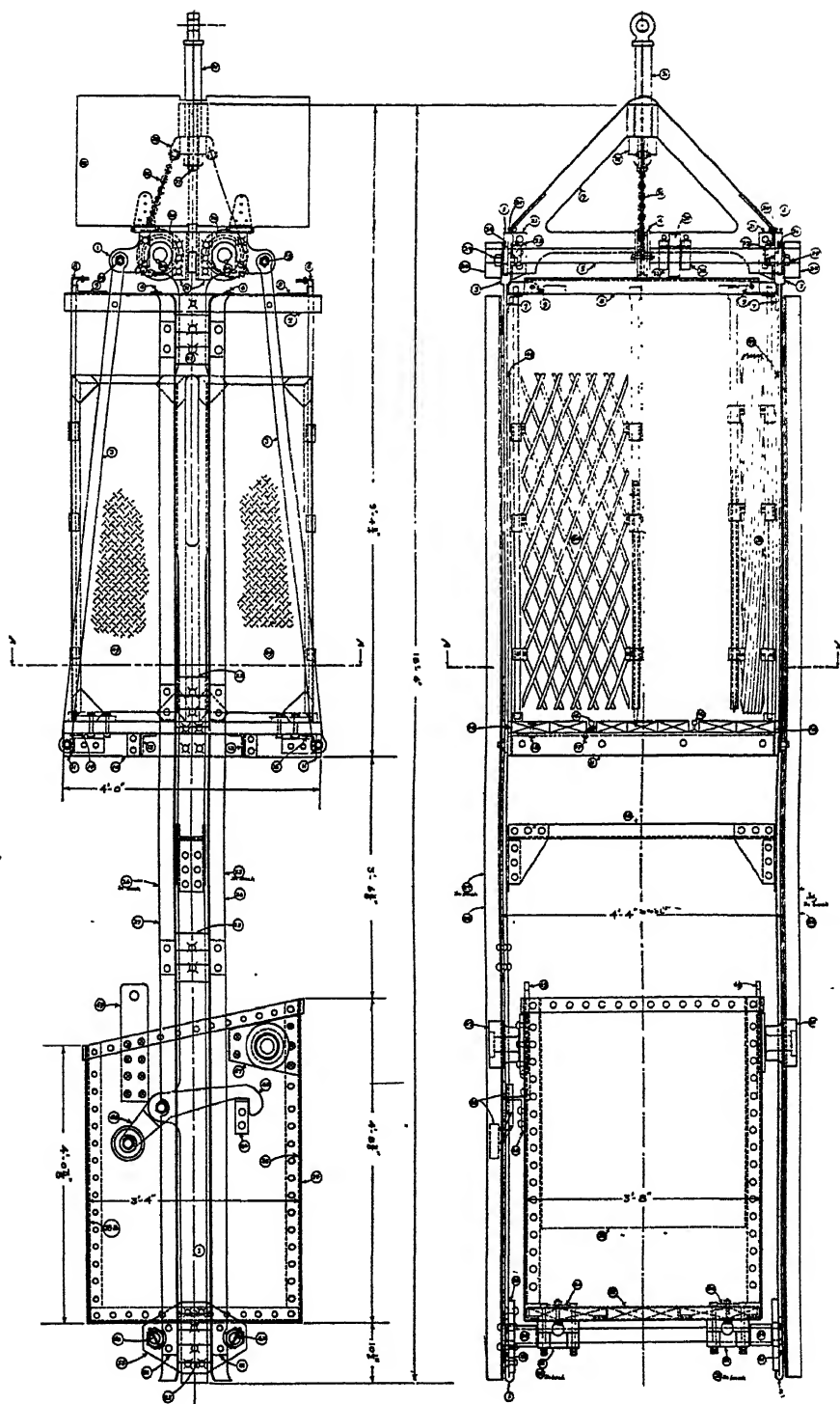


FIG. 8.—ASSEMBLY OF CAGE AND 2-TON SKIP, BIG JIM MINE. (Continued.)

permitted. Air was distributed to the stopes through 1-in. pipes in alternate manways.

Water for drilling purposes was distributed through 2-in. and 1-in. pipes on levels and $\frac{3}{4}$ -in. pipes in alternate manways.

Air Drills

Ingersoll-Rand No. 248 hammer drills were used for drifting. CC11 stopers of the same make and Sullivan DP33 rotators were used in stopes, the latter for blockholing.

Sampling

The mine foreman and shift bosses would take grab samples from broken ore in stopes and cut samples from development faces when making their first round after going on shift. Assay results would be in the foreman's office by noon on each day for samples taken during that morning and the night before. These results did not enter into any calculations whatsoever.

All tonnages and valuation calculations were based on carefully cut channel samples. Lengths of sample cuts were dependent on widths of bands of varying grades of ore in the vein, but never exceeded 10 ft. Face samples were cut with chipping hammers and back samples with light-weight stopers.

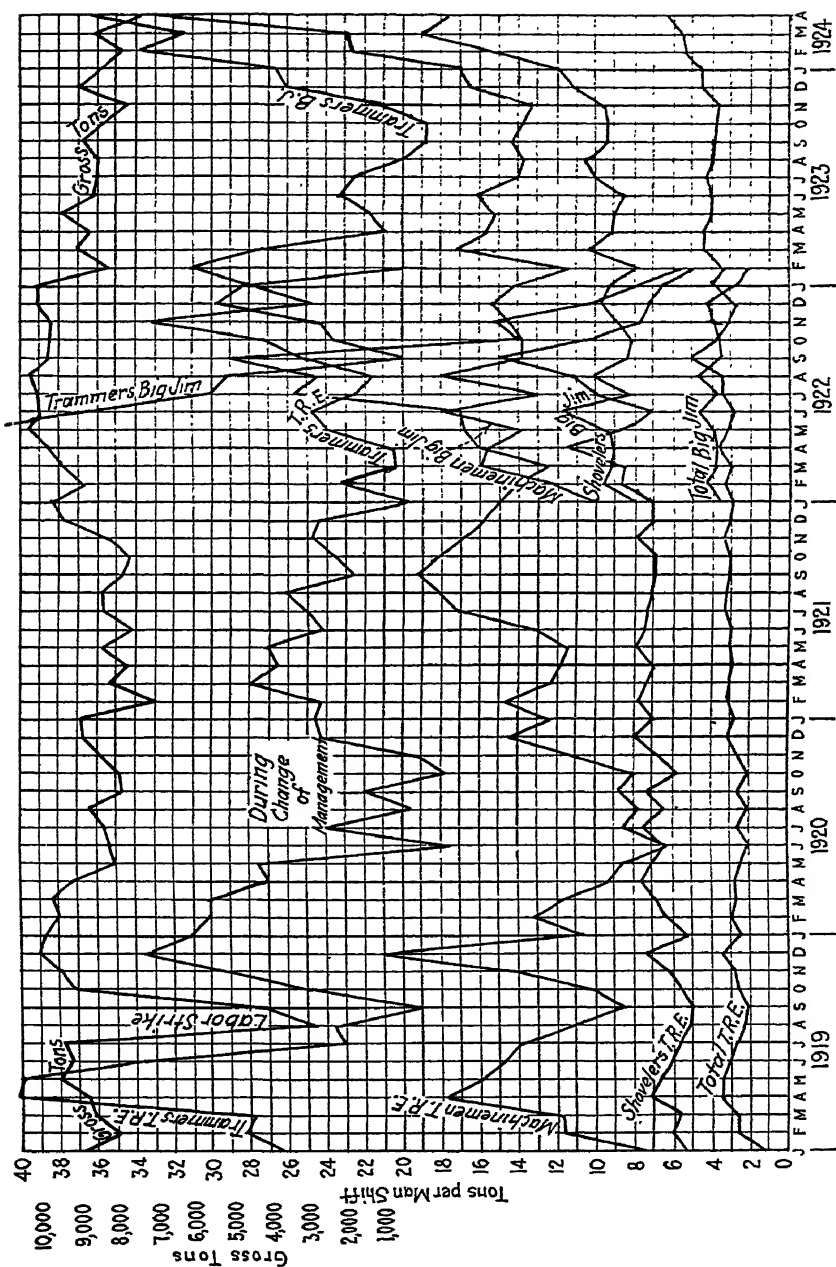
Tests indicated that carefully hand-moiled samples were more accurate than those cut by machines, but the latter method was more rapid and over a long period the percentage of error was insignificant.

Development drifts were sampled at 5 to 10-ft. intervals. Stopes were sampled at 10-ft. intervals, generally after alternate cuts in the back; *i. e.*, after each 12 ft. of advance.

For a given block of ore in the Tom Reed Extension body, it was determined that the grade as delivered to the mill was approximately 13 per cent. lower than its calculated value, with a corresponding or like increase in tonnage. This factor for the Big Jim orebody was approximately 16 per cent., the difference apparently being due to the fact that the gold was more irregularly distributed in the latter vein, all of which was not of commercial grade.

TRAMWAY

Ore was transported from the Big Jim mine to the mill over a Riblet aerial tramway equipped with 35 buckets of 10 cu. ft. capacity. The loads averaged 0.52 tons each. The terminal ore bins were 5080 ft. apart and the buckets were discharged at an elevation 8 ft. higher than



the loading point. The speed of the traction rope was 500 ft. per min. and the buckets were spaced 300 ft. apart. The loaded and return-track cables were respectively $1\frac{3}{8}$ in. and $\frac{7}{8}$ in. in diameter and of smooth-coil construction. The traction cable was $\frac{5}{8}$ in. in diameter and of 6 by 7 lang-lay construction. All cables were made of crucible steel.

Power input varied from 0 to 9 hp., depending on the position of a loaded bucket with respect to a breakover tower situated on a small hill midway between the terminals.

The tramway was operated by two men, one at each end, and they had no difficulty in delivering 300 tons of ore in 8 hours.

PRODUCTION AND COSTS

Tables 1 to 6 give production and cost figures for the full period of operation of United Eastern Mining Co. at Oatman.

TABLE 1.—*Production and Costs during Operation of United Eastern Mining Co. at Oatman, Arizona*

	AMOUNT	PER TON BASED ON COMBINED TONNAGE	PER TON BASED ON INDIVIDUAL MINE TONNAGE
Gross value ore milled	\$14,558,210.36	\$19.874	
Loss in tailing	495,065.02	0.676	
Values recovered.....	\$14,063,145.34	\$19.198	
Operating costs: (Exclusive of depletion, depreciation of plant, prepaid development, federal income taxes and litigation expenses).			
Mining—T. R. Ext.—direct	\$2,416,255.33	\$ 3.299	\$4.719
Mining—Big Jim—direct	757,004.99	1.033	3.432
Shaft sinking, T. R. Ext.....	15,993.42	0.022	
Milling—direct.....	1,473,424.63	2.011	
Marketing.....	89,481.39	0.122	
Indirect costs.....	1,036,440.56	1.415	
Sinking and development, No 3 and Big Jim shaft.....	86,090.96	0.118	
Labor, hauling, power and miscellaneous items in erecting tramway, additions to mill, etc. (Does not include costs of salvable material or equipment)...	45,511.03	0.062	
	\$5,920,211.61	\$ 8.082	
Income from direct operations.....	\$8,142,933.73	\$11.116	
Miscellaneous income.....	17,941.34	0.024	
	\$8,160,875.07	\$11.140	
Administrative expense.....	143,705.69	0.196	
Net income from operations.....	\$8,017,169.38	\$10.944	

Year	Tons			Gross Value			Operating Costs	Tailing Loss	Values Recovered T. R. E.
	T. R. E.	B. J.	Total	T. R. E.	B. J.	Total			
1917	84,548		84,548	\$1,895,805.32		\$1,895,805.32	\$615,801.00	\$68,134.82	\$1,827,670.50
1918	92,339		92,339	2,138,417.27		2,138,417.27	797,760.07	65,158.98	2,073,258.29
1919	97,325		97,325	2,031,594.20		2,031,594.20	896,574.12	61,085.04	1,970,509.16
1920	102,926		102,926	2,239,711.40		2,239,711.40	942,261.09	71,639.14	2,168,022.26
1921	95,807	1,906	97,413	1,952,441.11	\$21,149.20	1,973,590.31	943,858.50	63,536.53	1,889,640.58
1922	37,321	80,366	117,887	491,947.39	1,151,961.15	1,643,908.54	852,110.67	68,455.28	471,628.11
1923	1,710	103,090	104,800	20,689.70	2,064,384.83	2,085,074.53	723,343.75	91,905.35	19,732.90
1924		35,490	35,490		566,633.17*	566,633.17	274,266.67	21,624.26	
	511,976	220,552	732,528	\$10,770,606.39	\$3,804,128.35*	\$14,574,734.74	\$6,045,975.96	\$511,589.40	\$10,420,461.80
				Tailing retreated.....		16,524.38		16,524.38	
						\$14,558,210.36		\$495,065.02	

Year	Values Recovered		Gross Value per Ton			Costs per Ton	Losses per Ton	Net Income	Net Inc. per Ton	Per Cent. Extra.
	B. J.	Total	T. R. E.	B. J.	Total					
1917		\$1,827,670.50	\$22.423		\$22.423	\$7.283	\$0.806	\$1,211,869.41	\$14.334	96.41
1918		2,073,258.29	23.158		23.158	8.639	0.706	1,275,498.22	13.813	96.95
1919		1,970,509.16	20.874		20.874	9.212	0.627	1,073,935.04	11.035	96.99
1920		2,168,022.26	21.760		21.760	9.155	0.896	1,225,761.17	11.909	96.80
1921	\$20,413.20	1,910,053.78	20.379	\$13.169	20.206	9.678	0.652	966,195.28	9.930	96.78
1922	1,103,835.15	1,575,453.26	13.181	14.334	13.968	7.241	0.581	723,342.59	8.146	95.84
1923	1,973,436.28	1,993,169.18	12.099	20.025	19.896	6.902	0.877	1,269,825.43	12.117	95.59
1924	545,008.91*	545,008.91*		15.966*	15.966*	7.728	0.609	270,742.24	7.629	96.18
	\$3,642,683.54*	\$14,063,145.34	\$21.037	\$17.248	\$19.896*	\$8.254	\$0.698	\$8,017,169.38	\$10.944	96.49
					0.022		0.022			
					\$19.874*		\$0.676			96.60

* Includes values in mill cleanup. Exclusive of mill cleanup: Big Jim, 1924, 14,510; 1917-24 16,380; all ores; 1917-24, 19,303.

TABLE 2.—*Tonnage Produced during Operation of United Eastern Mining Co. at Oatman, Arizona*

Year	Tons Mined			Tons from Dumps			Total Tons Produced
	Tom Reed Ext.	Big Jim	Total	Tom Reed Ext. No. 1 Shaft	Big Jim	Total	
1917	84,548						84,548
1918	90,949			1,390			92,339
1919	97,325						97,325
1920	99,225			3,701			102,926
1921	95,807	1,606					97,413
1922	37,321	80,366					117,687
1923	1,710	101,491			1,599		104,800
1924		27,234			8,256		35,490
Totals.....	506,885	210,697	717,582	5,091	9,855	14,946	732,528

TABLE 3.—*Analysis of Indirect Costs for Operation of United Eastern Mining Co. at Oatman, Arizona*

Year	Superintendence		Cottage Expense		Offices Expense		Legal Expense		Taxes	
	Total	Per Ton	Total	Per Ton	Total	Per Ton	Total	Per Ton	Total	Per Ton
1917	\$ 7,197.16	\$0.085	\$ 1,497.94	\$0.018	\$ 8,506.66	\$0.101	\$ 1,866.55	\$0.022	\$15,800.21	\$0.187
1918	17,884.60	0.194	955.68	0.010	13,117.85	0.142	5,406.86	0.059	63,550.72	0.688
1919	14,770.94	0.152	766.10	0.008	10,934.91	0.112	5,674.06	0.058	97,613.09	1.003
1920	11,688.01	0.114	1,390.09	0.014	13,973.80	0.136	5,515.42	0.054	96,307.57	0.935
1921	9,778.44	0.100	876.93	0.009	14,885.16	0.153	6,709.47	0.069	117,545.78	1.207
1922	10,643.68	0.090	692.01	0.006	14,057.30	0.119	29,136.16	0.248	76,888.09	0.653
1923	9,084.22	0.087	1,183.62	0.011	13,338.55	0.127	23,676.07	0.226	47,949.12	0.458
1924	7,935.00	0.224	1,197.93	0.034	8,976.99	0.253	9,111.88	0.257	16,857.41	0.475
Total	\$88,982.05	\$0.121	\$8,570.30	\$0.012	\$97,791.22	\$0.133	\$87,096.47	\$0.119	\$532,511.99	\$0.727

Year	Liability Expense		Accident Expense		Fire Insurance		Miscellaneous		Total Expense	
	Total	Per Ton	Total	Per Ton	Total	Per Ton	Total	Per Ton	Total	Per Ton
1917	\$12,487.78	\$0.148	\$ 1,355.35	\$0.016	\$ 2,277.23	\$0.027	\$ 3,996.93	\$0.046	\$ 54,985.81	\$0.650
1918	18,963.21	0.205	553.60	0.006	3,039.11	0.033	8,288.84	0.090	131,760.47	1.427
1919	27,182.65	0.279			3,661.50	0.038	7,214.83	0.074	167,828.08	1.724
1920			31,505.64	0.306	3,518.63	0.034	7,748.08	0.075	171,647.24	1.668
1921			4,030.04	0.041	3,511.39	0.036	7,086.86	0.073	164,424.07	1.688
1922			3,280.03	0.028	4,036.18	0.034	6,400.00	0.055	145,133.45	1.233
1923			11,774.89	0.112	3,866.50	0.037	12,464.04	0.119	123,337.01	1.177
1924			5,258.95	0.148	2,946.59	0.083	25,039.88	0.705	77,324.73	2.179
Total	\$58,633.64	\$0.080	\$57,758.50	\$0.079	\$26,857.23	\$0.037	\$78,239.46	\$0.107	\$1,036,440.86	\$1.415

TABLE 4.—Mining Costs during Operation of United Eastern Mining Co. at Outman, Arizona

Year	Tons	Labor	Timber	Explosives	Other Supplies	Power	Miscellaneous	Total Cost
1917	84,548	\$ 203,808.16	\$ 51,834.82	\$ 41,938.78	\$ 20,312.17	\$ 21,313.52	\$ 5,306.50	\$ 353,514.04
1918	92,339	260,252.10	30,665.87	59,358.05	39,402.98	28,415.10	9,378.55	430,532.65
1919	97,325	307,440.46	48,007.65	62,362.09	30,416.59	28,031.09	12,723.19	494,981.97
1920	102,926	331,667.58	63,776.47	57,504.09	27,185.04	29,581.08	21,681.40	534,489.92
1921	97,413	249,344.13	48,402.02	45,766.15	31,733.05	40,351.41	11,837.35	427,431.11
1922	117,687	260,191.52	45,555.40	53,081.82	26,832.89	33,364.86	47,805.45	400,832.03
1923	104,800	220,184.17	29,306.66	39,511.96	22,277.20	21,324.19	38,805.48	371,529.66
1924	35,490	58,037.51	4,977.05	8,350.13	4,706.56	5,352.05	6,382.62	87,895.92
Total	732,528	\$1,890,925.03	\$831,580.03	\$307,963.07	\$218,017.08	\$207,737.80	\$156,980.60	\$3,173,210.30
1917		Per Ton	Per Ton	Per Ton	Per Ton	Per Ton	Per Ton	Per Ton
1918		\$2.410	\$9.613	\$3.496	\$2.317	\$2.252	\$8.063	\$4.181
1919		2.818	0.430	0.643	0.427	0.308	0.102	4.728
1920		3.158	0.404	0.641	0.374	0.288	0.131	5.086
1921		3.222	0.620	0.560	0.261	0.288	0.240	5.191
1922		2.560	0.497	0.459	0.326	0.421	0.122	4.388
1923		2.211	0.387	0.451	0.228	0.284	0.406	3.967
1924		2.101	0.280	0.377	0.213	0.203	0.371	3.515
		1.636	0.140	0.235	0.135	0.151	0.180	2.177
Average		\$2.581	\$9.453	\$3.502	\$2.208	\$2.284	\$2.214	\$4.332

TABLE 5.—*Analysis of Mining Costs, United Eastern Mining Co., Oatman, Arizona*

	Labor	Timber	Explosives	Other Supplies	Power	Miscellaneous	Total	Cost per Ton
Foreman and shift bosses.....	\$ 107,724.61					\$ 83.33	\$ 107,807.94	\$0.150
Sloping.....	887,103.57	\$314,763.23	\$199,817.58	\$ 82,358.99	\$ 62,523.30	3,920.01	1,550,495.77	2.161
Filling.....	368,527.18	4,132.15	90,754.95	39,090.93	35,463.77	2,406.14	516,395.12	0.762
Development.....	180,989.55	11,494.77	69,693.74	25,472.27	22,866.12	1,189.24	311,705.69	0.434
Dead work.....	10,014.47	1,071.63	375.76	372.25	416.45	8.29	12,258.85	0.017
Diamond drilling.....	23,653.04	104.25	1,321.04	8,386.84	7,719.29	7,196.59	48,381.05	0.067
Holisting.....	132,273.91			10,211.65	35,863.97	19,292.01	197,641.57	0.275
Dewatering.....	19,257.04			3,484.74	12,903.93	1,974.63	37,620.34	0.052
Assorting ore.....	11,017.09			163.04		0.07	11,180.20	0.016
Sampling.....	7,454.64			839.03	0.05	111.36	8,405.08	0.012
Assaying.....	16,672.30			6,661.41	1,850.05	325.01	25,509.37	0.036
Engineering.....	18,036.60			1,359.20	0.15	1,553.36	20,949.31	0.029
Ventilation.....	9,820.39			9,509.59	21,131.86	680.71	41,142.55	0.057
Aerial tramming.....	12,136.33			730.38	1,387.64	29,556.35	43,810.70	0.061
Miscellaneous.....	81,117.97			28,917.04	5,088.16	81,259.94	190,383.11	0.274
Total mining.....	\$1,885,798.69	\$331,586.03	\$367,963.07	\$217,557.36	\$207,215.43	\$149,566.07	\$3,150,686.65	\$4.403
Cost per ton mined.....	\$2.628	\$0.462	\$0.513	\$0.303	\$0.289	\$0.208	\$4.403	
Handling dump ore.....	5,126.94			489.72	522.37	7,414.62	13,553.65	0.907
Cost per ton from dump.....	\$0.343			\$0.033	\$0.035	\$0.496	\$0.907	
Total cost.....	\$1,890,925.63	\$331,586.03	\$367,963.07	\$218,047.08	\$207,737.80	\$150,980.69	\$3,173,240.30	\$4.332
Cost per ton.....	\$2.531	\$0.453	\$0.502	\$0.298	\$0.284	\$0.214	\$4.332	

MILL CLEANUP

The final mill cleanup is ably described in the report of the mill superintendent, E. M. Bagley, as follows:

By the 12th of May, 1924, the tonnage of ore supplied by the mine was so far below the mill capacity that work was started preparatory to cleaning up the plant. The outside agitators were emptied and the pulp worked through the remaining agitators.

The last of the ore was cleaned from the bins and the grinding department shut down on May 31. As soon as the pulp in each tank had been agitated a sufficient length of time it was transferred to the thickening circuit, agitators 7, 6, 5 and 4 being emptied in the order given. No. 3 thickener was then cut from the circuit and its contents pumped to No. 4 thickener. The pulp in agitators 1, 2 and 3 was then pumped to the thickening circuit, and thickeners 1, 2, 4, 5 and 6 emptied, and the solutions precipitated.

In emptying tanks, it was necessary to dump the contents upon the concrete floor, sluice the pulp to the sump, and pump it from there to the tank to which it was to be transferred. After all the tanks were emptied of pulp, the final cleanup of the unwashed pulp on the floor was accomplished by pumping the mud into No. 6 thickener and using the overflow for sluicing. The solution was then precipitated. In this manner we were able to collect the greater part of the dissolved gold in the solution to be precipitated.

The ore that had accumulated under the ore bins and conveyors was cleaned up and milled before the plant was shut down. There was a total of 73.75 tons cleaned up having an average value of \$10.34 per ton. Fifteen tons having a value of \$15.81 per ton were cleaned from around the mill storage tank, where leakage and overflow had enriched the ground.

We found that a concentration of gold had taken place in the hard muck below the rakes in some of the agitators and in the center of No. 1 thickener. Agitators 4, 5 and 6 were cleaned before the ball mills were shut down, so we were able to retreat the sand in the mill. This material did not run as high as that obtained from the first three agitators, but was of sufficient value to return more than the cost of retreatment.

The following are the tonnages and values per ton of the materials retreated:

AGITATOR	TONS	PER TON
No. 4	16	\$10.85
No. 5	14	6.41
No. 6	30	4.75

The sand below the rakes in agitators 1, 2, 3, 1A, 2A, 3A, and in the center of No. 1 thickener was sent to a local custom mill for treatment.

TABLE 6.—*Analysis of Mining Costs, United Eastern Mining Co., Oatman, Arizona*

Year	Labor	Timber	Explosives	Other Supplies	Power	Miscellaneous	Total
STOPPING							
1917	\$120,306.49	\$ 51,450.45	\$ 28,295.46	\$14,737.93	\$10,046.70	\$ 44.70	\$ 221,881.82
1918	113,366.71	38,018.90	25,498.55	13,066.48	7,021.49	50.10	202,001.23
1919	141,210.08	43,417.49	30,083.86	12,418.01	9,013.32	7.86	238,151.02
1920	184,502.25	62,325.83	27,517.93	8,841.50	7,746.88	452.42	261,170.85
1921	106,795.52	43,187.72	23,916.83	10,201.98	9,269.34	78.43	133,503.82
1922	112,250.16	41,850.66	31,256.61	10,000.86	10,231.24	1,252.43	207,511.96
1923	105,816.48	27,090.94	27,146.20	10,414.51	7,352.35	1,888.05	180,100.13
1924	27,766.88	4,811.55	6,012.12	2,077.69	1,816.98	221.12	42,730.34
Total.....	\$887,103.57	\$314,763.23	\$109,317.53	\$82,358.99	\$92,523.39	\$3,920.01	\$1,530,495.77
	Per Ton	Per Ton	Per Ton	Per Ton	Per Ton	Per Ton	Per Ton
1917	\$1.423	\$0.608	\$0.335	\$0.174	\$0.119	\$0.001	\$2.660
1918	1.301	0.418	0.280	0.144	0.077	0.001	2.221
1919	1.451	0.467	0.309	0.137	0.093	0.000	2.447
1920	1.558	0.628	0.278	0.089	0.078	0.005	2.636
1921	1.096	0.443	0.246	0.105	0.095	0.001	1.986
1922	0.954	0.356	0.266	0.090	0.087	0.011	1.764
1923	1.010	0.264	0.250	0.099	0.070	0.018	1.720
1924	1.020	0.177	0.222	0.076	0.067	0.008	1.570
Average.....	\$1.236	\$0.439	\$0.279	\$0.115	\$0.087	\$0.005	\$2.161
FILLING							
1917	\$ 33,328.12		\$ 8,882.91	\$ 4,371.23	\$ 2,751.33	\$ 11.61	\$ 49,345.20
1918	51,555.69	170.87	19,328.27	7,080.08	4,727.61	15.90	82,879.32
1919	71,863.48	937.43	19,010.74	7,853.28	5,394.86	8.23	106,069.02
1920	81,571.47	214.57	21,631.54	6,670.04	9,196.81	938.31	120,222.74
1921	48,781.31	2,589.75	14,815.62	5,803.34	6,957.13	4.42	78,651.57
1922	36,080.92		6,612.58	3,684.39	2,861.82	502.51	49,722.22
1923	41,402.94	239.53	5,869.26	3,665.01	2,338.78	899.94	54,415.46
1924	3,963.25		604.03	262.66	235.43	24.22	5,089.59
Total.....	\$368,527.18	\$4,152.15	\$96,754.95	\$39,090.93	\$35,463.77	\$2,406.14	\$546,395.12

	Per Ton	Per Ton	Per Ton	Per Ton	Per Ton	Per Ton	Per Ton
1917	\$0.304	\$0.002	\$0.105	\$0.052	\$0.033	\$0.000	\$0.584
1918	0.667	0.010	0.212	0.078	0.052	0.000	0.911
1919	0.738	0.010	0.195	0.081	0.060	0.000	1.090
1920	0.822	0.002	0.218	0.067	0.063	0.010	1.212
1921	0.501	0.027	0.152	0.057	0.071	0.000	0.808
1922	0.307	0.002	0.056	0.032	0.024	0.004	0.423
1923	0.395	0.002	0.056	0.035	0.022	0.009	0.519
1924	0.146		0.022	0.010	0.008	0.001	0.187
Average.....	\$0.514	\$0.006	\$0.135	\$0.055	\$0.040	\$0.003	\$0.762
DEVELOPMENT							
1917	\$ 10,645.04	\$ 384.37	\$ 4,760.41	\$ 2,389.48	\$ 1,619.06	\$ 18.53	\$ 10,816.89
1918	38,000.15	1,470.10	14,531.23	8,041.11	4,406.31	10.68	60,657.58
1919	35,327.83	1,652.76	13,267.40	5,093.76	3,382.53	1.47	58,725.81
1920	28,774.13	980.45	8,368.16	1,696.85	2,801.73	191.48	37,902.80
1921	14,670.42	1,774.64	5,591.05	2,095.17	2,092.77	6.91	26,232.16
1922	36,032.20	3,534.56	14,074.02	3,614.17	5,023.20	472.27	61,550.11
1923	10,376.54	1,526.49	6,486.50	1,770.82	1,503.56	318.35	28,028.26
1924	6,064.24	165.50	1,703.98	761.61	956.87	189.55	9,791.75
Total.....	\$180,080.55	\$11,494.77	\$60,693.74	\$25,172.27	\$22,806.12	\$1,189.21	\$311,705.69
	Per Ton	Per Ton	Per Ton	Per Ton	Per Ton	Per Ton	Per Ton
1917	\$0.126	\$0.005	\$0.056	\$0.028	\$0.019	\$0.000	\$0.234
1918	0.419	0.016	0.160	0.088	0.050	0.000	0.733
1919	0.363	0.017	0.136	0.052	0.035	0.000	0.603
1920	0.240	0.010	0.084	0.017	0.029	0.002	0.382
1921	0.151	0.018	0.057	0.022	0.021	0.000	0.269
1922	0.306	0.030	0.127	0.031	0.051	0.001	0.519
1923	0.156	0.015	0.062	0.017	0.014	0.003	0.267
1924	0.223	0.006	0.063	0.028	0.035	0.005	0.360
Average.....	\$0.252	\$0.016	\$0.067	\$0.035	\$0.032	\$0.002	\$0.431

TABLE 6.—*Analysis of Mining Costs, United Eastern Mining Co., Oatman, Arizona.—(Continued)*

Year	Labor	Timber	Explosives	Other Supplies	Power	Miscellaneous	Total
DEAD WORK							
1917	\$ 2,091.21	\$ 255.60	\$ 46.44	\$105.14	\$ 12.52	\$2.63	\$ 2,513.54
1918	5,740.22	776.39	328.42	220.90	331.87	1.39	7,399.29
1919	1,070.62	39.64	0.90	23.25	56.54	1.48	1,192.43
1920	1,064.25			22.70	15.49	2.78	1,095.22
1921	58.07			0.26	0.03	0.01	58.37
Total	\$10,014.47	\$1,071.63	\$375.76	\$372.25	\$416.45	\$8.29	\$12,258.85
	Per Ton	Per Ton	Per Ton	Per Ton	Per Ton	Per Ton	Per Ton
1917	\$0.021	\$0.002	\$0.000	\$0.001	\$0.000	\$0.000	\$0.025
1918	0.059	0.008	0.003	0.002	0.004	0.000	0.076
1919	0.009	0.000	0.000	0.001	0.000	0.000	0.010
1920	0.010			0.000	0.000	0.000	0.010
1921	0.002			0.000	0.000	0.000	0.002
Average based on entire tonnage produced 1917-1924	\$0.014	\$0.002	\$0.000	\$0.000	\$0.001	\$0.000	\$0.017
Average based on tonnage produced 1920-1924	\$0.022	\$0.003	\$0.000	\$0.000	\$0.002	\$0.000	\$0.027
Hauling							
1917	\$ 14,700.95			\$ 2,907.58	\$ 5,838.82	\$ 822.17	\$ 24,269.52
1918	13,163.67			1,096.87	9,225.31	1,130.89	24,616.74
1919	17,214.37			1,941.69	4,935.51	2,116.10	26,207.67
1920	20,323.73			1,250.36	4,363.70	396.06	26,333.85
1921	17,829.40			477.92	4,564.48	1,374.02	24,245.82
1922	27,261.09			1,238.82	3,767.51	6,475.13	38,742.55
1923	16,266.91			1,052.16	2,575.77	5,960.81	25,855.35
1924	5,514.09			246.25	592.87	1,016.86	7,370.07
Total	\$132,273.91			\$10,211.65	\$55,863.97	\$19,292.04	\$197,641.57

	Per Ton		Per Ton		Per Ton		Per Ton		Per Ton
1917	\$0.174		\$0.034		\$0.069		\$0.010		\$0.287
1918	0.145		0.013		0.101		0.013		0.271
1919	0.177		0.020		0.050		0.022		0.269
1920	0.205		0.012		0.014		0.001		0.265
1921	0.192		0.003		0.047		0.014		0.249
1922	0.232		0.010		0.032		0.055		0.239
1923	0.156		0.010		0.025		0.057		0.217
1924	0.202		0.009		0.022		0.037		0.270
Average.....	\$0.184		\$0.014		\$0.050		\$0.027		\$0.275
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DRAWERING									
1917	\$ 657.08		\$ 384.13		\$ 51.35		\$ 26.42		\$ 1,118.98
1918	2,727.07		734.54		204.98		0.24		3,660.83
1919	2,820.18		510.50		500.12		794.65		4,023.77
1920	4,273.53		300.79		936.07		119.02		5,635.41
1921	3,286.37		830.98		2,719.01		14.58		6,906.97
1922	3,518.12		388.12		3,778.99		313.46		7,908.99
1923	1,887.25		256.70		4,242.93		659.06		7,016.94
1924	87.44		16.38		470.41		47.19		621.45
Total.....	\$10,257.01		\$3,484.74		\$12,908.93		\$1,974.63		\$37,020.34
	Per Ton		Per Ton		Per Ton		Per Ton		Per Ton
1917	\$0.008		\$0.005		\$0.000		\$0.000		\$0.013
1918	0.030		0.008		0.002		0.000		0.010
1919	0.029		0.005		0.005		0.000		0.018
1920	0.043		0.003		0.010		0.001		0.057
1921	0.034		0.009		0.028		0.000		0.071
1922	0.030		0.003		0.032		0.003		0.068
1923	0.018		0.003		0.010		0.006		0.007
1924	0.003		0.001		0.017		0.002		0.023
Average.....	\$0.027		\$0.005		\$0.018		\$0.002		\$0.052

TABLE 6.—Analysis of Mining Costs, United Eastern Mining Co., Oatman, Arizona.—(Continued)

Year	Labor	Timber	Explosives	Other Supplies	Power	Miscellaneous	Total
ASSORTING ORE							
1917	\$ 992.00				\$ 14.10		\$ 976.10
1918	1,524.50				22.20		1,546.70
1919	1,552.80				28.48		1,581.37
1920	1,810.74				33.70		1,844.50
1921	1,335.70				33.74		1,369.44
1922	1,644.23				18.47	\$0.07	1,657.77
1923	1,545.15				13.00		1,558.15
1924	641.88				4.20		646.17
Total.....	\$11,017.09				\$163.04	\$0.07	\$11,180.20
	Per Ton				Per Ton		Per Ton
1917	\$0.012				\$0.000		\$0.012
1918	0.017				0.000		0.017
1919	0.016				0.000		0.016
1920	0.019				0.000		0.019
1921	0.014				0.000		0.014
1922	0.014				0.000	\$0.000	0.014
1923	0.015				0.000		0.015
1924	0.024				0.000		0.024
Average.....	\$0.016				\$0.000	\$0.000	\$0.016
SAMPLING							
1917	\$1,142.91				\$0.05		\$1,190.76
1918	432.34					\$ 0.24	462.77
1919	567.94					1.25	663.92
1920	564.12						609.43
1921	249.35					0.11	352.68
1922	2,174.67					27.73	2,404.58
1923	1,860.63					70.77	2,201.00
1924	462.68					11.26	510.94
Total.....	\$7,454.64			\$839.03	\$0.05	\$111.36	\$8,405.08

	Per Ton	Per Ton	Per Ton	Per Ton	Per Ton
1917	\$0.014	\$0.000	\$0.000	\$0.014	\$0.014
1918	0.005	0.000	0.000	0.005	0.005
1919	0.006	0.001	0.001	0.007	0.007
1920	0.005	0.001	0.001	0.006	0.006
1921	0.003	0.001	0.001	0.004	0.004
1922	0.018	0.002	0.002	0.020	0.020
1923	0.018	0.003	0.003	0.021	0.021
1924	0.018	0.001	0.001	0.019	0.019
Average.....	\$0.011	\$0.001	\$0.001	\$0.012	\$0.012
Assaying					
1917	\$ 1,870.14	\$ 914.68	\$ 31.84	\$ 3,011.66	\$ 3,011.66
1918	2,005.37	808.25	100.01	2,976.91	2,976.91
1919	2,072.97	635.83	32.12	2,749.12	2,749.12
1920	2,299.46	883.01	27.23	3,411.41	3,411.41
1921	2,452.83	1,023.25	548.79*	4,038.10	4,038.10
1922	2,638.32	1,122.64	783.40*	4,873.30	4,873.30
1923	2,331.50	838.18	302.21*	3,188.01	3,188.01
1924	692.05	215.27	10.93	960.11	960.11
Total.....	\$10,672.30	\$6,661.11	\$1,850.65	\$25,309.37	\$25,309.37
	Per Ton	Per Ton	Per Ton	Per Ton	Per Ton
1917	\$0.023	\$0.011	\$0.000	\$0.036	\$0.036
1918	0.022	0.010	0.001	0.033	0.033
1919	0.021	0.007	0.000	0.028	0.028
1920	0.023	0.010	0.000	0.031	0.031
1921	0.025	0.011	0.005	0.031	0.031
1922	0.025	0.010	0.006	0.031	0.031
1923	0.022	0.008	0.003	0.033	0.033
1924	0.023	0.009	0.001	0.035	0.035
Average.....	\$0.023	\$0.010	\$0.003	\$0.036	\$0.036

* High power costs 1921-22-23 due to large number of diamond-drill sludge samples dried on hot plates.

TABLE 6.—*Analysis of Mining Costs, United Eastern Mining Co., Oatman, Arizona.*—(Continued)

Year	Labor	Timber	Explosives	Other Supplies	Power	Miscellaneous	Total
ENGINEERING							
1917	\$ 1,633.75			\$ 59.78		\$ (7.47)	\$ 1,686.06
1918	2,133.06			61.26		0.65	2,197.87
1919	2,838.01			104.68	\$0.15	324.45	3,267.29
1920	2,143.61			116.59		305.38	2,565.58
1921	3,606.96			466.27		102.03	4,175.26
1922	2,303.64			248.63		150.01	2,702.28
1923	2,404.71			196.88		611.66	3,213.25
1924	969.96			105.11		60.65	1,141.72
Total.....	\$18,036.60			\$1,359.20	\$0.15	\$1,553.36	\$20,949.31
	Per Ton			Per Ton	Per Ton	Per Ton	Per Ton
1917	\$0.019			\$0.001		\$0.000	\$0.020
1918	0.023			0.001		0.000	0.024
1919	0.029			0.001	\$0.000	0.003	0.033
1920	0.022			0.001		0.003	0.026
1921	0.037			0.005		0.001	0.043
1922	0.020			0.002		0.001	0.023
1923	0.023			0.002		0.006	0.031
1924	0.036			0.004		0.002	0.042
Average.....	\$0.025			\$0.002	\$0.000	\$0.002	\$0.029
VENTILATION							
1917	\$ 631.99			\$ 597.71	\$ 410.16	\$ 3.07	\$ 1,642.93
1918	1,943.92			2,211.03	1,911.17	50.95	6,117.07
1919	2,555.06			4,830.99	3,269.75	31.80	10,687.60
1920	1,276.74			681.50	3,002.49	200.38	6,061.11
1921	1,689.79			710.34	5,725.93	0.22	7,426.28
1922	1,771.16			444.60	3,937.58	141.40	6,294.74
1923	500.07			28.21	1,573.03	216.24	2,317.55
1924	151.66			5.21	401.75	36.65	595.27
Total.....	\$9,820.39			\$9,509.59	\$21,131.86	\$680.71	\$41,142.55

	Per Ton	Per Ton	Per Ton	Per Ton	Per Ton	Per Ton
1917	\$0.007		\$0.007	\$0.005	\$0.000	\$0.019
1918	0.021		0.021	0.021	0.001	0.067
1919	0.026		0.050	0.034	0.000	0.110
1920	0.013		0.007	0.039	0.002	0.061
1921	0.010		0.007	0.059	0.000	0.076
1922	0.015		0.004	0.033	0.001	0.053
1923	0.005		0.000	0.015	0.002	0.022
1924	0.006		0.000	0.015	0.001	0.022
Average.....	\$0.014		\$0.013	\$0.029	\$0.001	\$0.057
ABRIAL TRAMMING						
1917						
1918						
1919						
1920						
1921	\$ 241.16		\$ 19.15	\$ 19.40		\$ 279.91
1922	4,604.05		400.19	519.70		18,196.82
1923	5,511.13		212.83	656.52		21,114.70
1924	1,770.30		38.71	192.73		4,219.27
Total.....	\$12,136.33		\$730.38	\$1,387.64	\$29,550.35*	\$43,810.70
	Per Ton		Per Ton	Per Ton	Per Ton	Per Ton
1917						
1918						
1919						
1920						
1921	\$0.003		\$0.000	\$0.000		\$0.003
1922	0.030		0.001	0.005		0.155
1923	0.053		0.002	0.006		0.202
1924	0.065		0.002	0.007		0.155
Cost per ton based on total tonnage produced in the years 1917-1924 inclusive.....	\$0.017		\$0.001	\$0.002	\$0.041	\$0.061
Cost per ton based on tonnage actually trammed 1921-1924.....	\$0.055		\$0.003	\$0.006	\$0.134	\$0.198

* Does not include \$12,779.43 written off as expense in 1921.
 Entire cost of tramway \$42,331.99 or \$0.1919 per ton (on 220,562 tons of Big Jim ore trammed).
 Entire operating cost, exclusive of depreciation = \$14,258.14 or \$0.0646 per ton trammed.

TABLE 6.—*Analysis of Mining Costs, United Eastern Mining Co., Oatman, Arizona.*—(Continued)

Year	Labor	Timber	Explosives	Other Supplies	Power	Miscellaneous	Total
MISCELLANEOUS							
1917	\$ 8,513.03			\$ 2,878.75	\$ 561.12	\$ 4,234.56	\$ 16,187.46
1918	10,192.82			6,224.60	728.10	6,151.79	23,600.40
1919	11,361.04			2,004.81	502.62	9,428.17	21,196.14
1920	12,271.93			4,587.89	507.10	9,467.81	26,324.85
1921	11,091.54			4,230.81	922.83	10,031.74	26,276.02
1922	11,374.78			4,285.69	950.04	25,816.81	42,136.32
1923	13,002.10			3,469.61	980.18	13,477.77	30,639.66
1924	3,310.73			735.45	216.99	2,348.26	6,611.43
Total.....	\$81,117.97			\$28,917.04	\$5,088.16	\$81,259.94	\$196,383.11
	Per Ton			Per Ton	Per Ton	Per Ton	Per Ton
1917	\$0.101			\$0.034	\$0.007	\$0.050	\$0.102
1918	0.112			0.069	0.008	0.071	0.200
1919	0.117			0.030	0.005	0.097	0.249
1920	0.124			0.042	0.005	0.090	0.267
1921	0.114			0.043	0.010	0.103	0.270
1922	0.097			0.037	0.008	0.219	0.361
1923	0.124			0.033	0.007	0.128	0.292
1924	0.121			0.027	0.008	0.086	0.242
Average.....	\$0.113			\$0.040	\$0.007	\$0.114	\$0.274
HANDLING DUMP ORE							
1917				\$ 22.47		\$1,680.83	\$ 2,576.68
1918	\$ 873.38			145.32		5,449.27	5,708.34
1919	113.75						
1920							
1921							
1922	653.58			61.26	\$ 74.34	27.67	816.85
1923	3,486.23			260.67	448.03	256.85	4,451.78
1924							
Total.....	\$5,126.94			\$489.72	\$522.37	\$7,414.62	\$13,553.65

	Per Ton	Per Ton	Per Ton	Per Ton	Per Ton	Per Ton
1917						
1918	\$0.020			\$0.016	\$1.200	\$1.851
1919	0.031			0.030	1.472	1.542
1920						
1921	0.409			0.038	\$0.047	0.511
1922	0.422			0.032	0.051	0.530
1923						
1924						
Average cost per ton based on tonnage actually taken from dumps, 1918, '20, '23, '24.....	\$0.343			\$0.033	\$0.496	\$0.907
Average cost per ton based on entire production 1917-24.....	\$0.072			\$0.007	\$0.103	\$0.189
FOREMAN AND SHIFT BOSS						
1917	Total				Total	\$
1918	9,407.00					9,407.66
1919	17,232.52					17,232.52
1920	18,056.61					18,056.61
1921	18,786.27					18,809.60
1922	16,535.97					16,535.97
1923	15,076.05					15,076.05
1924	9,541.17					9,541.17
	3,088.86					3,088.36
Total.....	\$107,721.61				\$83.33	\$107,807.91
	Per Ton				Per Ton	Per Ton
1917	\$0.111					\$0.111
1918	0.189					0.189
1919	0.186					0.186
1920	0.190					0.190
1921	0.169					0.169
1922	0.128					0.128
1923	0.091					0.091
1924	0.113					0.113
Average.....	\$0.150				\$0.000	\$0.150

TABLE 7.—Summary of Mill Cleanup, United Eastern Mining Co., Oatman, Arizona

	Tons	Values per Ton	Deductions	Total Values	Gross Values	Gross Deductions	Net Values
MILL SANDS TREATED AT CUSTOM MILL:							
Sand from thickener No. 1 and agitators exclusive of Nos. 4, 5, 6.....	97.341	\$ 88.79		\$ 8,635.03			
Sand from Marey feed box.....	4.606	22.53		103.77			
Slime from ball-mill floor, gutters and cone scale.....	2.425	26.04		63.15			
Classifier sand sent to smelter.....	5.255	483.58		2,541.66			
				<u>\$11,343.61</u>	\$11,343.61		
Custom mill deduction (8 per cent.).....			\$ 704.15				
Custom mill treatment charges.....			578.57				
Smelter deduction.....			143.84				
Smelter treatment charges.....			115.11				
Freight and hauling.....			179.34				
Labor and supplies.....			7,637.49				
			<u></u>	\$ 9,358.50		\$ 9,358.50	
				<u>\$ 1,985.11</u>			\$1,985.11
CHIP PILE CLEANUP:							
Sands treated at custom mill.....	64.951	\$ 36.05		\$ 2,521.41			
Ash treated at smelter.....	40.045	90.39		3,620.06			
				<u>\$ 6,141.47</u>	\$ 6,141.47		
Custom mill extraction deduction (8 per cent.).....			\$ 201.72				
Custom mill treatment charges.....			366.00				

[illegible]

There was a total of 97.341 tons of this material having an average value of \$88.79 per ton, or a gross value of \$8635.03.

After the ball mills were shut down, the classifiers and scoop boxes were cleaned and the material recovered was sent to the smelter. There were 5.255 tons of classifier sand assaying \$483.58 per ton, and having a gross value of \$2541.21.

The chips collected in the chip traps on the classifier discharges were saved from the time milling operations began. During the cleanup they were screened to separate the sand, and the chips were burned to ash. There were 69.951 tons of the sand averaging \$35.05 per ton and having a gross value of \$2521.41. There were 40.045 tons of ash resulting from the burning of the chips. This ash assayed \$90.39 per ton, and had a gross value of \$3620.36. The gross value of the chip pile was \$6141.47.

The refinery cleanup consisted of cleaning all precipitate lines and presses, the chipping of those parts of the concrete walls and floors that contained gold, the collection of dust from places of lodgment, the cleaning of the dust chamber and stack, and the recovery of shot and slug bullion from the furnace lining.

Tables 7 and 8 show the products recovered, as well as their values and costs of recovery.

TABLE 8.—*Sources of Bullion and Approximate Values*

Sources of Bullion	Pounds	Value per Lb.	Value per Ton	Total Values
Solution precipitated after all ore was ground.....				\$ 7,010.67
Scale from precipitation line from pump to refinery.....	527	\$39.65	\$79,310.81	20,898.40
Presses and precipitation line in refinery.....	196	28.71	57,428.56	5,628.00
Sweeps and chamber dust.....	158	5.70	11,398.00	900.00
Furnace lining bullion slugs.....				1,352.00
				\$35,789.07

Tailing Pond Recovery

Beginning with the year 1918, the tailing ponds were scraped to recover their surface accumulations of salts. Detailed costs for this work were not kept until 1920 (see Table 9). Expense includes cost of all labor and supplies properly chargeable to building and maintenance of ponds, scraping and hauling of salts to mill, and treatment.

Wind and rain were the two major factors that controlled the yearly recovery. The product is decreasing in amount and per ton value, but the recovery is still sufficient to return a portion of the wages of the plant watchman who oversees the work.

TABLE 9.—*Tailing Ponds Recovery, United Eastern Mining Co., Outman, Arizona*

Year	Dry Tons Tailing	Gold, Oz.	Silver, Oz.	Gold, Value	Silver, Value	Value per Ton	Total Gross Value	Costs			Net
								Expense	Royalty Paid	Total	
1920	153.887	274.383	211.558	\$ 5,672.00	\$ 220.02	\$38.22	\$ 5,892.02	\$ 4,530.56	\$135.55	\$ 4,672.11	\$1,219.91
1921	113.011	143.017	107.633	2,968.45	107.63	27.22	3,076.08	3,210.31		3,210.31	134.73 Loss
1922	173.131	258.051	143.040	5,331.38	143.04	31.03	5,477.42	3,720.76	160.73	3,887.49	1,590.93
1923	79.831	98.823	47.355	2,042.84	36.02	26.04	2,078.86	2,783.47		2,783.47	701.01 Loss
1924	424.865	468.731	459.346	9,689.32	308.40	23.53	9,997.72	6,368.25	362.95	6,731.20	3,266.52
1925	None										
1926	421.000	277.548	277.480	5,737.42	183.68	14.08	5,926.10	7,201.49		7,201.49	1,275.39 Loss
	1,365.725	1,521.153	1,216.412	\$31,444.41	\$1,003.79	\$26.70	\$32,448.20	\$27,827.31	\$659.23	\$28,486.57	\$3,061.63

The following is a typical analysis of the material retreated:

	PER CENT.		PER CENT.
Water.....	13.81	Sodium chloride.....	11.10
Silica.....	47.75	Iron.....	Trace
Calcium carbonate.....	23.47	Thiocyanates.....	Trace
Sodium carbonate.....	0.37	Cyanides.....	None as such
Sodium bicarbonate....	2.81	Alumina.....	Trace
Sodium sulfate.....	4.60		

DISCUSSION

G. A. PACKARD, Boston, Mass.—This paper covers the mining of two orebodies in western Arizona which were mined by the cut-and-fill method. This method was adopted after the orebody had been, as I understand it, fairly well delimited, and after a study of all of the factors that would have a bearing on net profits, a real broad engineering basis. The method adopted was not the one which appeals to the mine shift boss, the one which will give the greatest number of tons per man-shift, or the lowest cost per ton; it was the method that was deemed capable of yielding an ounce of gold at the lowest figure.

H. S. MIDD, Los Angeles, Calif.—One of the interesting points Mr. Moore does not touch upon is the comparison between the ultimate result of production and profits and the estimates that were made before the mill was erected. Walter Wiley and the company's own engineers thoroughly sampled the mine to the 6th level and made up a set of figures on tonnage and grade and estimates of cost. I regret that I have not Mr. Wiley's figures with me, to compare with the figures given by Mr. Moore. An absolute comparison could not be made because the company did not keep separate records of the ore mined above and below the 6th level but the net result was extremely close. The grade of ore and the tonnage ultimately produced agreed very closely with the estimates made before the mine was operated.

The mine was equipped and worked without departing from standard practice. Mr. Moore's paper presents a picture of the life history of a high-grade gold mine of medium size which was developed and worked under exceptionally favorable conditions; two oreshoots were exhausted and the history of the mine came to an end. Before profits ceased the lateral and vertical continuations of the vein were explored by underground workings and diamond-drill holes. The main oreshoot became impoverished below the 8th level and no other orebodies were discovered. After $7\frac{1}{2}$ years of operation the mine was closed down and its history came to an end without the long period of endeavor and hope to find other orebodies which so frequently happens in mines of this type and which sometimes results in the dissipation of profits which have been earned during the earlier life of the property.

J. A. BURGESS, San Francisco, Calif. (written discussion).—I have not read Mr. Moore's article on costs at the United Eastern mine, but I know pretty well what it contains. Mr. Moore is now with me at Copala, Sinaloa, Mexico, and we have talked over the article together. Inasmuch as I started the work at the United Eastern mine and carried it on for over four years, we thought that a few words from me would be of interest. Of necessity I must write without referring to any records.

Perhaps the most important move in starting the operation from a cost-of-mining point of view was the choice of the method of mining the orebody. Various methods were considered, including those of shrinkage, cut and fill, square-setting with later filling, and various methods used at Bisbee, described in a paper by Philip D. Wilson.⁵

⁵ P. D. Wilson: Stoping in the Calumet and Arizona Mines, Bisbee, Ariz. *Trans.* (1916) 55, 118.

I had just come from the Nevada Wonder mine in Nevada, where I had been using both shrinkage stoping and cut-and-fill stoping, and I was familiar with the square-set method and other methods used at Tonopah.

It had been expected, before my arrival, that the beautiful orebody of the United Eastern could be mined at a very low cost, but after studying the situation thoroughly I gave as my estimate a total cost of \$7 per ton for mining, milling and overhead. This was in the early summer of 1916, before the war had its full effect on mining costs. This figure was not far from the actual costs in the earlier months of operation, but the records show how the costs mounted during the war and again receded in the later years of operation.

My first hope was that the shrinkage method could be used, as this, if applicable, would have lowered the cost of mining very considerably, perhaps by \$2 per ton. The considerations that decided the question in favor of the cut-and-fill method were:

1. There was a heavy hanging wall in most parts of the mine due to the altered character of the andesite and to gouge seams parallel to the vein. It was obvious that there would be a serious slabbing of waste into the ore and even of extensive caving if the shrinkage method were used. This danger proved later to have been more serious than was anticipated, when a large block of the hanging wall, extending from below the 400-ft. level, began to slide on a fault seam into the stopes. It was, of course, held to a safe degree of movement by the filling. Even with the small spaces opened under the cut-and-fill system, it was often necessary to shore up the hanging wall with lagging to keep back the swelling gouge.

2. As a variation of the cut-and-fill method, it was hoped that a rill system could be combined, but in early operations, it was found, after the vein had been cut out for stoping over a considerable distance along the drift, that the vein had a tendency to split off in long horizontal blocks about 5 or 6 ft. thick. This was probably due to side pressure from the walls added to the overhanging weight of the ore. To avoid trouble from this source the back was supported with posts. This tendency to split in large horizontal blocks, and the consequent necessity for the supporting posts, was incompatible with the use of sloping backs or down-hanging points such as would be required by rill stoping or other somewhat similar methods that were considered. The posts also interfered with the use of a scraper in spreading filling, although not to the extent of preventing its use entirely.

3. Most of the filling was secured from inclined raises or waste stopes in the walls. This served another important purpose, beside that of making filling. It gave perfect assurance that no ore was being left in the walls, something that is difficult to do with the shrinkage system.

4. The high grade of the ore made it imperative that no chances be taken of losing ore by the caving of shrinkage stopes. Later events proved that there would have been serious loss from this cause if the shrinkage method had been adopted.

During my time in Oatman, the adjoining Tom Reed mine secured considerably lower stoping costs than the United Eastern, by using the shrinkage method, because their veins were narrower, the walls were stronger, and their conditions were adapted to the method used. Their ore, as I recollect it, assayed something like \$10 or \$12 per ton and they could afford to take greater chances with it. At times they mined \$8 ore.

In much of their mining they achieved a very low mining cost by giving the stoping and tramming to contractors at a per-ton rate. This again would not do with our \$20 to \$25 ore.

Two factors that increased our cost of mining over what it would have been with a low-grade ore from a narrower vein were (1) the high grade of the ore and the consequent care required to avoid losses in stoping, by admixture with the filling, or other wise, and (2) the width of the vein, which involved the necessity of supporting the

back while stoping. Narrow veins, with weak walls, can sometimes be worked by shrinkage by using stulls and headboards, but this cannot be done where the width runs up to 40 ft., as was the case in parts of the United Eastern vein.

Apropos of contracting the stoping, after I had left Oatman, I was called back as a witness in a law suit, and this involved a close examination of a shrinkage stope that was being mined at a per-ton rate. I noticed that the miner was idle for hours at a time, and while it was not my business, I asked him in a friendly way whether he was waiting for something. He knew that I had no official connection with the mines, and therefore confidentially told me that he was holding back on breaking tonnage to keep his profits down to a figure that would not challenge unfavorable attention and cause his rate to be reduced. Evidently there were still further possibilities in the contract system.

A. NOTMAN, New York, N. Y.—This paper is a most valuable contribution, presenting, as it does, a picture of the complete operations of the life of a mine, and giving not only cost but profits and all the essential data. We owe a debt of gratitude to those who supplied it.

In the tabulation of cost on page 77, under "Development," I presume the total amount did not include the development already performed at the time the property was acquired. It merely applies to what was done after acquisition.

S. JENNINGS, New York, N. Y.—I agree with Mr. Notman in his valuation of this paper. Mr. Mudd, who was largely instrumental in bringing about this development work, told us recently that it was possible for an organization to take hold of small mines and make them profitable if it had the courage to stop when it was through. This is an example of the sound business judgment of the men who are at the head of this enterprise, who were able and courageous enough to say that the profits that had been obtained should not be expended in hopeless, or almost hopeless search for further orebodies. This is an example that the engineers will have to take somewhat to heart. We are all, with our disposition as miners, full of optimism and hope, and we always think that just beyond some particular point there is going to be another orebody. We must use more conservative judgment in that respect.

G. A. PACKARD.—The point brought out by Mr. Jennings is certainly most important—not only as to stopping when we are through, but when we are exploring; to stop when we have done a reasonable amount of exploration.

B. F. TILLSON, Franklin, N. J.—I do not find any charge or amortization of capital investment which has been applied in the operating cost. It looks fine that out of a total growth value of some \$14,500,000, a net income of \$8,000,000 should be made from the operation, but the other limiting factor is that it would not be if \$10,000,000 had been invested. No amortization had occurred in the operating cost. So the story seems to me incomplete.

H. S. MUDD.—I do not recall the actual figure, but I believe the investment was less than \$500,000 for everything.

R. W. MOORE (written discussion).—A comparison between ultimate production figures and Mr. Wiley's estimates would be of great interest and value but, as Mr. Mudd points out, records were not kept in a way that would permit a segregation of tonnages produced from blocks embraced by Mr. Wiley's report. Any comparison would of necessity have to be based on map records of assays and vein widths rather than on tonnages of ore extracted and mill-head assays thereof. Such a comparison would be of doubtful value.

Comparison of Branch Raise and Combined Shrinkage and Caving Methods*

BY CHARLES A. MITKE,† PHOENIX, ARIZ.

(Los Angeles Meeting, September, 1928)

EXCLUDING top-slicing and sublevel caving, large production caving methods may be divided into two general classes, the branch raise, or undercut caving method, and the combined shrinkage and caving method, each of which has a number of variations, the result of continual efforts towards obtaining lower costs and better extraction, by the introduction and substitution of minor improvements and modifications.

EARLY CAVING METHODS

Present caving methods are the cumulative result of developments covering a period of many years, with the most rapid growth during the past 15 years. They originated, presumably, from early English stoping methods, and, as far as is generally known, were first introduced in the Michigan iron district. They have had their greatest expansion and application in the large low-grade porphyry coppers. In their present form, they are probably a development and combination of sublevel caving and shrinkage (back stoping on ore) with gravity handling of broken material.

Sublevel caving, as practiced in the Michigan district today, is a combination of drift, or crosscut, mining, with top-slicing, and caving, supplemented by later refinements in the shape of mechanized handling of broken ore. A height of anywhere from 16 to 20 ft. is generally taken, and the ore is drilled and blasted along the drift or crosscut for half this height (say 8 to 10 ft.), after which the upper 8 to 10 ft. caves and breaks of its own weight, with comparatively little assistance from drilling. This

* The writer was requested to prepare a comparison of so-called "block" caving methods, the intent being to compare caving methods other than top-slicing, and sublevel caving. He does not favor the term "block" caving, because it is too general and not sufficiently expressive of the methods to which it is applied. All underground mining is "block" mining, for whatever the extraction method used, a "block" of ground is usually prepared for stoping operations. It may be a square "block," a rectangular "block," or an irregular shaped "block," but, in a wide sense, it is a block, or section of the orebody. For this reason, the term "block" caving has been omitted from the title of this article, and the words "branch raise, and combined shrinkage and caving methods" substituted.

† Consulting Mining Engineer.

upper 8 to 10 ft. is naturally broken at a much cheaper cost than the lower 8 to 10 feet.

The first attempts at caving on a large scale in the Michigan district, known there as "block" caving, were a radical departure from sublevel caving systems, in that the height of ore caved was increased, from a former maximum of 8 to 10 ft., to a total of 100 to 125 ft. The originators and introducers of this variation were undoubtedly spurred on by a desire to reduce the comparatively high cost of sublevel caving, by increasing the height of ore broken by caving, after the lower 8 to 10 ft. had been mined out by drift, crosscut, or top-slice mining.

In this particular instance, the block of ore (approximately 250 ft. square and 125 ft. high) was undermined by a series of parallel drifts and crosscuts, which left the block standing on fairly large pillars (22 ft.). It was further weakened by carrying up narrow shrinkage stopes around the boundaries of the block on several sides. When this was accomplished, the pillars were drilled and blasted, allowing the block to settle the height of the drifts (approximately 8 ft.), which started cracking and caving in the mass above. One objection to this early caving method was that in order to facilitate removal of the broken ore, heavily timbered drifts, using spiling, had to be driven through the caved mass, and considerable hand shoveling of ore into cars was necessary. While the general idea of undercutting the block was a step in the right direction, no control of the caving ore was possible, and the recovery of broken ore from stopes proved costly.

As the method was improved and given a wider application at the early Utah copper, the Ohio copper, and later, the porphyry copper deposits of the West, it was found advantageous to locate the main haulage drifts a considerable distance (70 to 100 ft.) below the block being caved. An undercutting level, composed of a series of parallel drifts, was driven in the bottom of the block of ore. Between this level and the main haulage level beneath, an intermediate level was driven, known as the grizzly, or breaking level. This was usually 15 or 20 ft. below the undercutting level, and 50 to 80 ft. above the main haulage level, so as to provide chute room. Many small raises served to carry the ore, broken and caved on the undercutting level, to the grizzly level, where large boulders were blasted before they passed through the grizzly bars into the large chutes comprising the ore pass system, which conveyed the broken ore to the cars on the main haulage level below. This system of large incline raises, branching out into smaller raises, served to expedite the passage of broken ore by gravity, and did away with the old hand-tramming and shovel methods.

Practically every large caving system today makes use of an ore pass system for the handling of broken ore. This, of course, involves a considerable expense for raising, but has been found more economical than

the early hand work, providing the height of ore caved justifies the expense. Practically the same amount of development put in for a height of 16 to 20 ft. will serve a height of 150 to 250 ft. In a few instances, branch raise ore pass systems have been applied to sublevel caving, but, in general, the limited height of the ore caved makes their use prohibitive.

IMPORTANT VARIATIONS IN LARGE-SCALE CAVING METHODS

As time passes, the caving methods, as practiced on a large scale, at the low-grade deposits, are becoming less and less distinctive and are gradually merging into one another, so that, at present, probably their greatest divergence consists in the amount of drilling and blasting required to break the different classes of ground and the manner of drawing the ore once it is broken.

Hard ground requires considerably more drilling and blasting than medium or soft ground. Certain classes of ore will not cave readily if merely undercut, but require more breaking and blasting. In such ground, variations of the well-known room-and-pillar method are used.

The-room-and-pillar method is, of course, one of the oldest methods in use. In early mining, and likewise today, in certain of the harder deposits, the stope ore is broken by drilling and blasting, and the pillars are either allowed to remain in place, or, after the stope ore has been extracted, the rooms (or stopes) are filled with waste and the pillar ore mined at a later date. Where shrinkage is combined with caving, the stope ore is broken by drilling and blasting—merely the swell being drawn out—and, when the stopes are full, the pillars are undercut (or broken in some other manner) and simultaneous drawing of both stope and pillar ore commences. The back, being unsupported, gives way and caves, following the withdrawal of the ore.

In the branch raise, or undercut caving method, as applied to the medium and soft ores, the block of ore to be mined is not divided into rooms and pillars, but, as previously described, is merely undercut by a series of parallel drifts, which may be either on the horizontal or rilled. From these drifts, drill holes are put up at various angles, which, when blasted, cause the block to drop the height of the drift, and start the ore running. The entire block continues cracking, breaking, caving and descending towards the chutes, and a continuous flow of ore is thus established. Caving and breaking of the block is facilitated by carrying up small shrinkage stopes on three and occasionally four sides, which isolate it from the rest of the mass, and prevent the arching effect that is the result of undue support of the upper portion of the undermined block. One of the greatest drawbacks to an efficient caving system is to have the block arch and "hang up," as this introduces many complications such as excessive dilution with waste, air blasts, etc.

Where shrinkage stoping must be used, probably from 50 to 75 per cent. of the ore must be drilled and blasted, before drawing commences, whereas where the block is merely undercut only approximately 5 to 10 per cent. of the ore is broken. The quantity of powder required also varies and with the former method may range from 0.54 to 0.70 lb. per ton of ore, while with the latter it may be as low as 0.16 and 0.28 lb. Naturally, the latter method should be, and is, the cheaper of the two, as less labor in drilling and a smaller quantity of explosives in blasting are required. In both cases, hand-tramming of broken ore is eliminated, and the ore runs by gravity into cars on the haulage level below.

The type of ground to be mined is the determining factor in basing a conclusion as to which of these methods should be employed. Hard ground does not cave as readily as medium and soft ground, and is apt to break in massive blocks, which are unwieldy to handle. However, it has been found recently that by a more general use of boundary shrinkage stopes much ore, which was formerly classed as "hard," can be made to cave, merely by breaking the block with shrinkage for only a portion of the height instead of the entire vertical distance.

GRIZZLY LEVELS AND ORE PASS SYSTEMS

Generally, both the undercut branch raise caving and the combined shrinkage and caving methods use an intermediate level, located between the bottom of the block being caved and the main haulage level. This level, which is generally driven about 15 to 20 ft. beneath the shrinkage stopes, and approximately the same distance beneath the undercutting level (branch raise system), is known as the grizzly, or breaking level. Here, large boulders are broken up before they pass into the transfer chutes, thus reducing the annoyance of chutes "hanging up," and facilitating the flowage of ore. Early caving methods did not make use of this supplementary level. The Ray mines operated without one for nearly 10 years, but where large daily tonnages are handled it has been found that greater efficiency, a more continuous flow of broken ore and fewer delays ensue when large boulders, which have a tendency to block chutes, are broken before they pass through the grizzly bars. Wherever the height of ore to be caved warrants the expense, grizzly levels are now in use at Ray.

The Creighton mine at Sudbury (International Nickel Co.) does not use a grizzly level. The ore, which is fairly hard, is broken by shrinkage stoping and simultaneous pillar caving. Very large raises, or "box holes" as they are locally termed, are used, and the largest boulders are blasted in these. The ore then runs to an underground crusher, which reduces all rock to 8 in. or less. Arguments are advanced by operators both for and against the use of grizzly levels, but, in general, it will be found that the grizzly level has a wide application.

Where shrinkage is employed to break the ore, there is a question whether to blast the ore fairly small in the stopes and merely break up large boulders on the grizzly or to do only a small percentage of breaking in the primary blasting operation in stopes and most of the breaking on the grizzly in the secondary blasting. The ore has to be broken up somewhere and opinions vary as to where it should be done. The Ray mines break most of their ore moderately small in stopes; Alaska Juneau, on the contrary, does by far the larger percentage of its breaking and consumes most of its powder (probably 80 per cent.) on the grizzly level. One reason for this is that, whereas the Ray ore is broken from below (the old method of back stoping on ore), Juneau ore is broken from above, the men coming down long raises from the level above, and setting off large powder blasts in short drifts. Once production commences, no one enters the stope, consequently they have no alternative but to break the ore on the grizzly level. The same applies to La Touche mine (Beatson, Kennecott), where the ore is also broken from above, the men putting in long drill holes from the bottom of inclined raises. These are, of course, merely minor variations in the method of breaking ore in shrinkage stopes.

DRAWING ORE FROM CAVING STOPES

The second important variation in the caving methods under discussion is in the manner of drawing the broken ore. It is one thing to start a mass of ore caving, breaking and running through innumerable outlets below, and quite another to control the ore flow so as to obtain certain results at will.

In the branch raise, or undercut caving system, the mine organization has removed the supports from beneath a comparatively large block of ground, thus bringing into play the forces of gravity. Cavities are formed as ore is drawn through the chutes and nature, which abhors a cavity, commences to fill them with whatever material is most accessible, regardless of whether it is ore or waste. The efforts of the organization are directed towards recovering the ore and allowing the waste to remain behind. Avoidance of dilution of ore with waste depends to a large extent on keeping the waste overburden intact, and lowering or caving it in an even and regular manner. If it is too badly cracked and broken, the waste, which is often finer than the ore, may find its way through cracks, chinks between large boulders, and funnels, or pipes in the caving ore, and soon reach the chutes, where it will intermingle with the ore being drawn out and lower the metallic content.

Ideally, the overburden should be brought down horizontally, but there are many obstacles to such an outcome. It has been found in practice that if an angle of 60° away from the center of operations is maintained—that is, if the overburden inclines down towards the extreme end of the orebody at such an angle, and drawing is practiced by retreat-

ing towards the solid—a very large percentage of the ore can be recovered clean, or free from dilution with waste. It is impossible, of course, to get up into the caving mass and see what is happening above the draw holes, but experience has taught that the greater the number of draw holes, and the closer they are spaced together, the more even will be the draw (providing the right amount of ore is drawn from each point) and the greater the recovery.

With shrinkage stopes, where access to the stope has been possible, and the ore drilled and blasted from beneath, the mass has been broken into boulder of fairly regular size and, consequently, drawing from such stopes usually results in a slightly better extraction than where the mass is broken through caving and crushing.

Where shrinkage is used, generally two incline raises serve to conduct the ore from the stope to each grizzly. These latter are spaced at regular intervals along the grizzly level.

In the branch raise, or undercut caving method, variations exist in the manner of conveying the broken ore to the grizzly level. At Inspiration, Miami, and the Andes Copper Co., two incline raises lead up from each grizzly towards the undercutting level. However, midway between these two levels, square sets are built in each of these incline raises, and smaller, so-called “finger raises” driven from the sides of these square sets to tap the undercutting level. Therefore, the ore passing through any one grizzly is first drawn through eight small finger raises, then through the two larger incline raises which conduct it to the grizzly, where the boulders are broken up and the entire mass passed through the grizzly bars into the large transfer chute of the ore pass system. These large transfer chutes beneath the grizzly level are in turn branched so as to serve two or more grizzlies.

The square sets in the incline raises above the grizzly level are known as “draw points.” Controls are placed in each of the four finger raises leading from them and men are stationed in the square set. It is their business to draw equal amounts of ore from the finger raises in rotation, so as to facilitate the even lowering of both ore and waste capping. Each of the draw points, or square sets, controls the extraction from a certain small block of ground directly above. There are a number of these draw points, and when drawing takes place from each square set and finger raise in regular order, the entire block of ore is gradually brought down. The men soon become very accurate in gaging the amount of ore drawn from each finger raise, and close supervision is also exercised from the grizzly level by the engineering department.

There are variations in the way the square set, or drawing point is located, that is, the angle at which it is set, which also determines the inclination of the finger raises leading from it. Inspiration sets the square set one way; Miami, another. At the Ruth mine (Nevada Consolidated

Copper Co.) the square set is built over the grizzly and all drawing is done from this point. As the ground is heavy, a regular grizzly drift is not driven and the top of the transfer raise is merely widened out sufficiently to contain the square set and grizzly. A manway is maintained in the transfer raise, or ore pass system, for access to the grizzly.

Several variations are also in effect at Morenci (Phelps Dodge Corp.). One consists of driving two short incline drifts from each grizzly and timbering them with drift timbers. A number of small finger raises are put up to the undercutting level from alternate points in the sides of these short drifts. These provide a greater number of drawing points than when square sets are used. The man is in the drift, and draws from each raise in turn. In another part of the mine, where the ore is softer, it is allowed to cave right down over the top of the drift, and no finger raises are used at all. These variations in the method of drawing exist by virtue of the fact that special merits, in the form of lesser dilution and easier drawing ore, are claimed for each by their sponsors.

More important than the type of finger raise, or square set used as a draw point, is the spacing of the draw holes where they tap the broken

TABLE 1.—*Spacing of Draw Holes at Various Mines*

Name of Property	Spacing of Draw Holes, Center to Center, Ft.	Area Served by Each, Sq. Ft.
Inspiration Consolidated Copper Co.	12½ by 12½	156¼
Miami Copper Co.		
Andes Copper Co.		
Eastern Iron Mines.		
Nevada Consolidated Copper Co.		
Ray mines, hand-tramming method.....	12½ by 20	250
Ruth mine (Nevada).....		
Nevada Cons. Copper Co.	14 by 14	196
Ray mines, motor-haulage method.....		
Phelps Dodge Corpn.	15 by 15	225
Morenci mines, combined shrinkage and caving..		
Bisbee district.....	12½ by 25	312½
Another variation in use in Arizona mines.....		
	Grizzlies staggered along drifts.	
International Nickel Co.	30 by 50	750
Creighton mine.....		
	Box holes staggered along drifts.	
Kennecott Copper Co.	40 by 60	1050
La Touche (Beatson).....		
Alaska Juneau Gold Mining Co.....	42 by 80	1680

ore above, and the area served by each. If these openings are too far apart, the ore lying between is apt to form an inverted cone, with pipes, or mill holes, directly over the draw points, through which the upper and more distant ore is drawn, while the piled up rock on the sides remains unmoved. Naturally, this reduces the total recovery, and also increases dilution. On the other hand, there is a limit to the close spacing of these openings, as there is considerable wear and tear on the sides of the finger raises, due to the passage of large quantities of broken ore, and if the rock forming the walls of the finger raises is soft, and the openings are too close together, they are likely to wear through and merge into one large glory hole, thus defeating the purpose for which they are intended.

Table 1 gives an idea of the spacing of the draw holes, or tops of drawing raises, at the various properties, where the combined shrinkage and caving, and branch raise, or undercut caving methods are used.

The amount of ore served by a group of finger raises or draw holes, controlled from a square set or given drawing point, is usually estimated before drawing commences. Assuming that a ton of ore is equivalent to $12\frac{1}{2}$ cu. ft., each block, $12\frac{1}{2}$ by $12\frac{1}{2}$ ft. square, and 1 ft. high, would contain $12\frac{1}{2}$ tons. When the height of ore to be caved is known, the contents of the block can readily be estimated. This is generally known as the "expectancy." A record is kept of the cars of ore drawn from each of the blocks served by the control points, and a comparison of the totals of these two will give a good idea of the recovery obtained.

MINOR VARIATIONS IN CAVING METHODS

In addition to those already described, there are a number of minor variations.

In the shrinkage stope and simultaneous pillar-caving method (combined shrinkage and caving), there are variations in the width of stopes and pillars used (these depending on the character of the ground); likewise, in the height of ore mined in any one lift. If the ore breaks and runs well, a considerable vertical height may be taken. On the contrary, if the ore is wet, sticky or "packy," tends to pipe or clog in the chutes, it is better to take shorter lifts.

Then there is the question as to the manner of breaking the ore, whether to use stopers, and break the ore in the old-time fashion of back stoping on ore, drilling the rock overhead; to take it off in horizontal slabs, using *drifters*, or to break it in steps or rills. It may also be broken in a more general way, by setting off large powder charges, placed either in long drill holes or in powder drifts driven for that purpose. Some companies use steel of ordinary length while others prefer 16 to 20-ft. lengths. Naturally, this blasting breaks the ore into large boulders, which must be broken up a second time, either in the stope, on the grizzly,

or, if no grizzly is used, in the large chutes leading from the stopes. The choice of a location for this secondary breaking gives rise to other minor variations. There are also variations in the location of grizzlies, some being placed in the center of the drifts, others along the sides, alternating, first one side and then the other, or in other words, staggered, while still others are placed in short offset drifts.

In the branch raise, or undercut caving system, the minor differences consist in whether the undercutting level is horizontal or rilled; in the vertical distance between the undercutting and grizzly and the grizzly and haulage levels; and also in the horizontal spacing between the parallel drifts that comprise these levels, which, in turn, to a large extent governs the spacing of draw holes. Variations in the location of the square sets, or drawing points, and the finger raises tributary to them, have already been described. There are also minor differences in the manner in which these square sets are timbered. The ordinary square set is used in some cases, while in others, additional timber supports in the shape of the letter A are used.

COST OF CAVING SYSTEMS

The caving methods (exclusive of top slicing and sublevel caving) are all low-cost methods. As applied at certain properties, some cost less than others, depending largely on the character of the ground, how close the draw holes are spaced (which governs the amount of raising required), and the amount of care and supervision that must be exercised in drawing to keep the dilution with waste at a minimum.

The problems of fill, support of back, or timbering (except for the draw points and certain sections of the drifts) do not enter into the selection of a large production caving method for a new orebody, the fundamental points involved being the way in which it is proposed to break the ground, and, having broken it, how it shall be drawn, so as to permit the highest recovery of the metal with the least amount of dilution, the cost of extraction, of course, taking into consideration the value of the metal content.

Where it is agreed at the outset to accept dilution of the ore with waste, large tonnages can be broken at one time, and very little supervision need be exercised over the draw. Consequently, a very low mining cost is possible. Alaska Juneau, on account of the characteristics of the ore and surrounding waste, accepted dilution in the selection of the caving method in use. The mine produces 10,000 tons daily, the ore being broken by large powder blasts, somewhat similar to those used in quarries and railroad cuts (21,000 lb. of powder set off at one time resulted in the breaking of 250,000 tons of ore, which proved to be a key block and caused the subsequent caving of 3,000,000 tons of ore a few hours later).¹

¹ A Modern Way of Breaking Ground. *Engng. & Min. Jnl.* (1928) 125, 546.

Neither close spacing of draw holes nor close supervision of drawing is exercised, and a remarkably low mining cost of about 25 c. (including supervision) is obtained, but 56.8 per cent. of the tonnage mined is sorted out as waste, before milling.

At Ray, on the contrary, most of the ore is hard and must be drilled and blasted by shrinkage, in some parts of the mine from 50 to 75 per cent. of the ore being broken in this manner. Close spacing of draw holes is practiced, as otherwise the ore pipes and a small total recovery would result, and close supervision of drawing is also necessary to maintain the grade. The mining cost for 1927 was 0.7696 c. per ton² (inclusive of all overhead but excluding depreciation, taxes and prepaid development), with a dilution of probably 5 to 10 per cent. on a daily production of 8000 tons.

Miami, with a class of ore that lends itself far more readily to caving, needs only to undercut the mass, drilling possibly anywhere from 5 to 10 per cent. of the ground. Close spacing of draw holes is practiced and close supervision of drawing is enforced. As a result, a mining cost of 41 c. (exclusive of general expense) is obtained,² with a dilution of between 5 per cent. and 10 per cent. on approximately 12,000 tons daily.

ORGANIZATION AND SUCCESSFUL ADAPTATION OF CAVING METHODS

For a caving method to be a complete success, every man in the organization must be trained to do his part efficiently. Men should be tried out at different jobs and then assigned to the work to which they are best adapted. Some may have special aptitude for driving haulage drifts, others for driving small grizzly drifts, while still others are better at raising, and this work is again subdivided into large and small raises. Some men show special efficiency in timbering stopes, others in timbering drifts. Altogether, there are at least 12 distinct classes of work necessary in preparing a block of ground for caving. The lowest cost per ton in preparing a block of ground for caving has been obtained where contract prices have been figured out for these different classes of work. A low wage scale for such work does not always imply a low total cost. Frequently, better results are obtained where the men receive higher wages through contract. This applies equally to deposits of very low grade. Therefore the working out of a proper contract or bonus system and the selection and training of the right men for each particular class of work are important duties of the organization.

At one large low-grade property one crew of two to six men drives the large 8 by 8 ft. haulage drifts. These are followed by a timber crew. Another crew drives the smaller 5 by 7 ft. grizzly and undercutting drifts; still other crews put up the raises and install the grizzlies, and so

² Annual Report for 1927.

on through the different classes of work necessary in the preparation of the block for drawing. These crews are rarely interchanged, as they are made up of men who have been selected because of their aptitude for their particular kind of work. Instead of a straight wage of \$5.00 the men earn from \$5.50 to \$8.00 on contract, and the company also benefits because the work of preparation is done more quickly and cheaply than when done on the regular day's pay. When the work is finished in one block of ground, the men are transferred to another.

After a block of ore is prepared for drawing, the same care should be exercised in selecting men to control the draw. These "chute tappers" should have special aptitude for sizing up volumes, as they must gage the amount drawn from each finger raise solely by the time involved in drawing, and by personal observation of the tonnage drawn. Some men can do this more readily than others. They must also know how to displace boulders in finger raises, in order to expedite ore flowage and avoid delays, when these raises frequently become choked. The shift bosses and engineers who supervise the draw should also receive training, so that they can accurately supervise the volume of ore coming from each raise. They should be able to take tonnage figures from the ore cars below and proportion them over the different finger raises above, so as to make sure that the capping is drawn down evenly. Accuracy and promptness are two necessary qualifications for both the chute tappers and the supervisory force.

At most camps contract and bonus prices are paid only for development work, but several companies are expanding these to include the stoping and are obtaining very good results. The bonus is based on the tonnage produced and the assay value of the ore mined, and includes not only the men in stopes but also those on the tramming level, and the supervisory force as well. In this way a premium is paid on quality as well as quantity.

In conclusion it may be stated that the successful operation of the caving systems depends not only on a selection of the right method, but also very largely on the proper application of that method by the mine organization.

DISCUSSION

F. W. MACLENNAN, Miami, Ariz.—So much has already been written on the various details of block caving that I do not think I can add anything of general interest. We have done a great deal of mining at Miami by various caving methods and it might be worth while to call attention to some of the features that are important in connection with that operation.

To begin with, it is of prime importance to size up carefully the orebody from the point of view of hardness or strength or toughness, degree of fracturing, etc., in order to determine in advance as accurately as possible the areas which it would be necessary to undercut in order to promote caving, and which at the same time would not result

in excessive weight and excessive maintenance charges. The system at present used at Miami was developed primarily to meet the necessity of obtaining a very low cost which was imperative on account of the very low grade of the ore, which is less than 0.8 per cent. sulfide copper. As a considerable proportion of the total mining cost is the cost of sinking the shafts, driving the haulage levels and transfer raises, grizzly levels, ore-drawing raises and the actual preliminary mining of the ore to promote caving, it follows that if this work can be applied to a high lift of ore rather than a shallow lift, the expense will be spread over a larger tonnage and in this way largely reduce it, and with this object in view it was decided to undercut and draw lifts of ore at Miami averaging approximately 300 ft. thick, which has resulted in attaining an over-all cost, including development and delivery to the crusher plant of 38.76 c. per ton for the 11,947,178 tons of ore mined to date. This cost is very close to the estimated figure before mining operations were started. The ore is being mined at the rate of approximately 12,000 tons per day.

As pointed out, the undercutting and caving of high lifts of ore reduces the cost considerably, but on the other hand, it increases the difficulties of control of ore drawing to avoid admixture of waste and to avoid throwing excessive weight on the extraction openings. To control the ore drawing, it is necessary that the ore be made to cave to predetermined boundaries. With low lifts of ore, this has been reasonably well accomplished in the past by putting up narrow shrinkage stopes along these boundaries, but with high lifts of ore this practice would result in throwing excessive weight on the extraction levels and probably completely wrecking them. To take care of this at Miami, boundary caving drifts were driven around the entire boundary of each stope, which is 150 ft. wide by 300 ft. long, at vertical intervals of 30 ft. This vertical interval was later changed to 45 ft. and the backs of the drifts shot in. The former interval had the effect of weakening the boundary plane by one-quarter and the latter by one-third. This method causes the back of the stope to hang up and become stressed to such an extent that the tendency is for it to break up into small fragments; this is very much to be desired and prevents the dropping of the entire back of the stope as a solid plug, which happens where the stope is completely cut off around the boundaries by shrinkage stopes. When this happens the inevitable result is the wreckage of the extraction openings below.

The Miami orebody would be termed weak ground; if it is undercut over an area of, say, 150 by 300 ft., or even 150 ft. square, it will ultimately cave. Sometimes this occurs immediately and sometimes the back of the stope will hang up for a month or more until the daily dropping of small amounts of ore from the back culminates in a general cave-in. This is usually accompanied by evidence of more or less weight on the timbering in the extraction openings below.

The present method of mining at Miami has been in operation for three years, with very satisfactory results, and it is apparently well adapted to this particular orebody.

F. AYER, Morenci, Ariz.—Our system of caving at Morenci is more or less similar to that being used at Miami and Inspiration, except that we draw from the grizzly level instead of from a pony control set between the grizzly level and the undercut level. We have an ore which is very much harder than the Miami ore; in fact, I do not know of any place in the world where harder ore is being caved than at Morenci. This involves certain changes in our method of mining. For instance, we cave by checkerboarding the undercut level into small blocks about 8 by 14 ft., and blasting a line or part of a line at a time. We cannot get very far ahead with our undercutting for fear we will drop an immense chunk which would throw undue weight on the grizzly level timber and be very difficult to re-undercut, break up, and pull down into the chutes. On account of having "big" ground the ore breaks in large pieces and

previously required an enormous amount of hammering to get it through the 10-in. grizzlies. To reduce labor costs we have widened the grizzly spacing to 16 and 18 in. This has resulted in a very marked increase in stoping efficiency. A year ago the total stoping efficiency was about 35 tons per man, while now it is over 80 tons per man, and we are getting about 125 tons per chute tapper. This increase is not due solely to wider grizzlies. There have been many other changes, such as improvement in timber design on the grizzly level, which has done away with a lot of repair work; scheduling of stope development work on and above the grizzly level so that only one crew operates in any grizzly drift at a time; the elimination of a large amount of shrinkage stoping, etc. However, the greater part of the increased improvement in efficiency has been due to the wider grizzlies.

There is danger in using a very wide grizzly if proper precautions are not taken. During the last month we have lost one man through a 16-in. grizzly. To do away with this danger we now have a safety belt on every chute tapper. To this belt a rope about 9 ft. long is attached and the other end is made fast to a cap 6 or 8 ft. back from the edge of the grizzly. This enables a chute tapper to jump back 12 or 16 ft. in case a large boulder rolls out to one side of the grizzly. It also makes it practically impossible for the chute tapper to fall through the grizzly into the chute.

When I was at the Moctezuma Copper Co. property we lost two men at our Pilares mine through 10-in. grizzlies, and at other mines where I have worked men have fallen through spaces less than 10 in., so you can see that grizzlies with most any spacing commonly used are hazardous if proper precautions are not taken.

I would like to ask Mr. MacLennan—Have you lost any men through 10-in. grizzlies?

F. W. MACLENNAN.—Not lately.

F. AYER.—Where at Miami they are using 150-ft. stopes, the stoping panels in Morenci are 112 ft. wide and may extend almost any length. Each panel is served by two haulage drifts 56 ft. apart, and ore is drawn off through double chutes spaced 28 ft. center to center along the length of these haulage drifts. This spacing was chosen so that ore may be drawn simultaneously from two or more chutes into 10-ton cars which are 14 ft. from coupling to coupling. Inclined raises are driven from motor level chute sets on both sides at 75° inclination to grizzly level 50 ft. above. Grizzly drifts are run over these raises at right angles to direction of haulage drifts. Grizzlies are either 14 or 18 ft. apart along grizzly drifts, and finger raises are driven out from each side of each grizzly to the undercutting level 20 ft. above the grizzly level. The raises are belled out ahead of or during undercutting, and a spacing of 14 by 18 ft. centers between raises is maintained.

Stoping is commenced by connecting on the undercut pilot raises driven along the panel boundary. The backs of these raises are blasted down and a shrinkage stope about 6 ft. wide is carried up to the capping or near enough to it so that the back will ravel up to the capping, and cutting off of the panel longitudinally is assured. It has been possible in sections where the ore caves more readily to carry shrinks up about 30 ft. and then omit shrinking a section of about 50 ft. in length and carry the shrink up on each side. This gives the longitudinal section a sawtooth appearance, and has resulted in quite a saving. Previously transverse shrinks were carried across the panels but this practice has been discontinued. Undercutting drifts are started as soon as panel is cut off by shrink on each side. These drifts are run over the finger raises from grizzly level at 14-ft. intervals and crosscuts are driven every 18 ft. The undercut is thus checkerboarded and as two or three 14 by 18 ft. squares are ready they are drilled with a standard fan-shaped round which breaks the ground between, and also breaks about 5 ft. over the drifts. As said before, it is not good to allow too big an area to be undercut. A smaller area and a longer time allowance for strains in the rock to be set up yield the smallest sized pieces of ore when caving starts. The

gangue, being quartz monzonite porphyry, presents some different problems in caving from that in Miami district, where the chalcocite is in schist, but on the whole caving in this hard ground is proving very satisfactory.

G. J. YOUNG, San Francisco, Calif.—As I understand it, the Miami mining method has been pretty well applied over a considerable area, but when the point has been reached where a given area has been completed with respect to alternate stopes, there are blocks or “pillars” between the worked out and caved blocks. I would like to ask Mr. MacLennan if work has been started on these in-between areas, and if so, how they have acted.

F. W. MACLENNAN.—We started mining the first pillar stope last fall, almost exactly one year ago. This stope contained approximately 1,100,000 tons and to date we have mined about 1,000,000 tons, or about 90 per cent. The results have not been quite as good as the average of the original stopes, either from the point of view of costs or extraction of metal. This was due to excessive weight in the pillar stope, which caused a large amount of maintenance and increased the cost, and to a large number of chutes off the ore-drawing list, which decreased the extraction.

Making use of the information gained in mining out the first pillar stope, the size of the second pillar stope was reduced from 150 by 300 ft. to 150 by 150 ft. This stope was started last Spring and 100 per cent. of the ore has already been mined with costs only slightly higher and metal extraction considerably better than the average of the original stopes.

In spite of the fact that we permit the capping which has followed the ore drawing down from the surface in the original stopes to settle for approximately a year, so that it will afford substantial support on the boundaries of the pillar stope, we must expect greater weight in mining out the pillar stopes but we have every confidence that this operation will present no considerable difficulty now that we have cut down the pillar stopes to one-half their original size, and we expect the extraction from these pillar stopes to be fully as good as the extraction from the original stopes, as the tendency of the ore, once it is thoroughly broken, is to draw down vertically and not from the sides, so we do not expect dilution from the waste filling in the original stopes.

The capping at Miami runs much more easily than the ore, because as a rule it breaks finer, so it is necessary to exercise care in the control of ore drawing. Our records to date show that 25 per cent. of the ore is drawn from a stope before capping first appears and 78 per cent. of the ore is drawn clean.

F. AYER.—Our capping, if anything, is harder than our ore, being highly silicified. It breaks into larger chunks than the ore, and for that reason we do not have trouble with piping, which properties in Miami have, as the finer the ore or the capping the greater the tendency to pipe and allow waste to flow in at the top of the pipe. With our coarser capping over rather coarse broken ore we do not have to guard against dilution from capping as do they at Miami.

F. H. PROBERT, Berkeley, Calif.—It would be unfortunate to inject at this time any academic discussion on so interesting a subject as “caving methods,” but for the benefit of the inexperienced, perhaps for the good of all who may have these problems of deciding upon mining practices presented to them, would it not be well for us to pronounce the alphabet and clearly define the terms we use? I have heard reference to “hard,” “very hard,” “soft,” “altered,” “highly silicified” and “porous” rocks. These words are relative, they do not mean anything unless you have a standard clearly in mind. The likelihood is that entirely different interpretations obtain in

different districts. We need a common yardstick with which to work in any descriptions of mining methods or comparisons of practices in different mining districts.

We have many standardized mining methods; perhaps too many. Nevertheless it is true that there are no two orebodies alike, and no two mineral deposits identical in character, physically or chemically. We cannot change a mineralized mass to suit our past experience; we must, then, in the development of new fields, looking to the improvement of mining practices, adapt our knowledge to conditions as we find them. I plead for a common language that we will all speak and can understand when describing these problems.

I remember reading the excellent papers written by G. Townsend Harley³ and Gyss and Davis⁴ dealing with the physical character of rock masses; a subject demanding most careful study. More such contributions are highly desirable.

There are but two kinds of rock, all -isms and -ologys to the contrary notwithstanding—*ore* and *waste*. The thing is to know the difference. As our knowledge of the way rocks behave under varying conditions of stress and strain, natural or induced, advances, the waste of yesterday may be the ore of tomorrow, but before much progress can be made either in investigation or exchange of experiences we must agree upon the definition of terms, and the language in which we express our thoughts must be intelligible to all.

E. D. GARDNER, Tucson, Ariz.—The United States Bureau of Mines is getting under way a comprehensive study of mining methods. Particular attention will be paid to the factors affecting costs and the reasons for the choice of different methods. With the cooperation of the operators, the A. I. M. E., and perhaps other agencies, I believe that results of value to the industry will be obtained. Without the active cooperation and support of the mining companies particularly, such an investigation could not be successful.

I also wish to second Dean Probert's plea for the adoption and use of terms that will have the same meaning in all parts of the country.

E. P. MATHEWSON, Tucson, Ariz.—I would like to have someone offer a definition of "hard rock." I realize that this is difficult to do.

F. W. MACLENNAN.—It is not easy to define the terms "hard," "soft," "tough," etc., as applied to ore or rock, as these would vary, depending on the point of view. For instance, from the point of view of drilling, a rock or ore might wear out the drill steel rapidly and be called hard, while, on the other hand, it might cave very readily when undercut, due possibly to thorough fracturing, and from the point of view of mining it might be termed soft in contrast to its terminology from the point of view of drilling.

Regarding the use of a grinding mill to test rocks for hardness, one rock might classify one way under coarse crushing and another way under fine grinding. The particular ore in the Miami mine, which is a highly silicified schist, is very easily crushed down to 4 mesh because it is very thoroughly fractured, but from this point down to about 60 mesh the fine grinding is much more difficult, on account of the siliceous character of the ore.

³ G. T. Harley: Proposed Ground Classification for Mining Purposes. *Engng. & Min. Jnl.* (1926) **122**, 368, 413.

⁴ E. E. Gyss and H. G. Davis: The Hardness and Toughness of Rocks. *Min. & Met.* (1927) **8**, 261.

Liquid-oxygen Blasting at Chuquicamata, Chile

By H. C. SCHULTZ* AND F. K. MIDDLETON HUNTER,† CHUQUICAMATA, CHILE

(New York Meeting, February, 1928)

CERTAIN local conditions were known to govern in large measure the successful adaptation of liquid-oxygen explosives to the large-scale blasting at Chuquicamata. The wide variation in hardness of the rock at this mine makes necessary the use of several types of explosives. A locally manufactured, relatively cheap black powder is used in blasting the softer waste and capping material, where a thorough displacement breaks the ground sufficiently. Before the introduction of L. O. X., an ammonia dynamite of about 65 per cent. strength was being used exclusively in the harder underlying orebody, where a strong disruptive as well as a displacement effect is required for breaking the ore. The L. O. X., as developed for quarry work in the United States, has a practically pure carbon cartridge material, and the explosive has an extremely high rate of detonation, which should therefore cause it to develop usefully its full disruptive effect in hard ground.

The individual blasts at Chuquicamata are large, black-powder shots which dislodge masses of rock averaging 250,000 tons each, and dynamite shots which dislodge masses averaging 80,000 tons each. It was desirable, therefore, to attempt to displace with L. O. X. the faster, higher-cost ammonia dynamite used in the smaller blasts. It was not considered practical to attempt to displace the low-cost black powder with L. O. X., as estimated costs showed no probable saving if this were accomplished.

Since it was desirable to continue shooting relatively large blasts for effective operation, it was recognized that larger storage facilities than had been in use would be needed for the liquid oxygen. Because of the difficulty of transporting and discharging large storage containers of efficient design, the better plan appeared to be storage and discharge of the liquid at the place of manufacture, and transportation of the filled soaking boxes to the blast.

PRESENT LIQUID-OXYGEN PLANT

An experimental unit with a rated capacity of 75 liters per hour was erected during the latter part of the year 1926 to prove the effectiveness of L. O. X. at this mine, its cost, and possible efficiency of large-scale storage of the liquid. With this experimental unit 200,000 tons of ore

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† Junior Engineer, Chuquicamata Mine, Chile Exploration Co.

is being blasted monthly; a total of 1,265,700 tons has been blasted with L. O. X. to Oct. 31, 1927.

The air is compressed to 853 lb. per sq. in. in a three-stage Ingersoll-Rand compressor. The high-pressure decarbonizing apparatus and the Claude system liquefaction apparatus were supplied by L'Aire Liquide, from Paris.

The flow sheet (Fig. 1) differs from that shown by O'Neil and Van Fleet¹ in two respects. In the first place, the decarbonizing or scrubbing of the air is accomplished after it has been compressed; this saves floor space and probably something in the initial cost of a plant, over the other method of scrubbing at atmospheric pressure before compressing. In the second place, the work done by the expansion engine on its piston is utilized to run a direct-connected surpressor which recompresses part of the main air circuit entering the liquefaction apparatus to 2845 lb. per sq. in. (for facilitating liquefaction by its expansion later), instead of converting this work into electric energy in a generator, or dissipating it as heat.

The expansion engine has been the cause of more operating delays than any other part of the liquefaction apparatus. It requires a relatively frequent changing of piston packing leathers, and frequent close adjustment of the valves.

Plant production is not quite the rated 75 liters per hour, but ordinarily about 73.5 liters per hour. It varies within limits determined by the perfect working of all machines; any wear on packing leathers, slight clogging of heat exchanger tubes, or freezing of the liquefying apparatus with ice or solid carbon dioxide immediately has its effect in decreasing production. The plant is operated 24 hr. daily, including Sundays, to keep storage losses at a minimum.

STORAGE OF LIQUID OXYGEN

The problem of storage on a large scale was submitted to the Purox Co. of Denver, which had had experience in the design of the smaller vacuum wall containers. Consideration was given to the insulating quality of vacuum and solid insulating material for large liquid-oxygen containers. L. E. Bedortha, of the Purox Co., estimated that a container holding 1000 gal. would require an insulating layer 10 ft. thick (of cork board, mineral wool or similar insulating material) to equal the insulating value of a 1-in. vacuum space (vacuum corresponding to an absolute pressure of 0.0001 mm. of mercury). This would mean a cumbersome storage tank 30 ft. high by 25 ft. diameter. Some disadvantages of such a container are:

1. Adequate support would conduct heat into the container.

¹ F. W. O'Neil and H. Van Fleet: Liquid Oxygen as an Explosive. *Trans.* (1926) 74, 690.

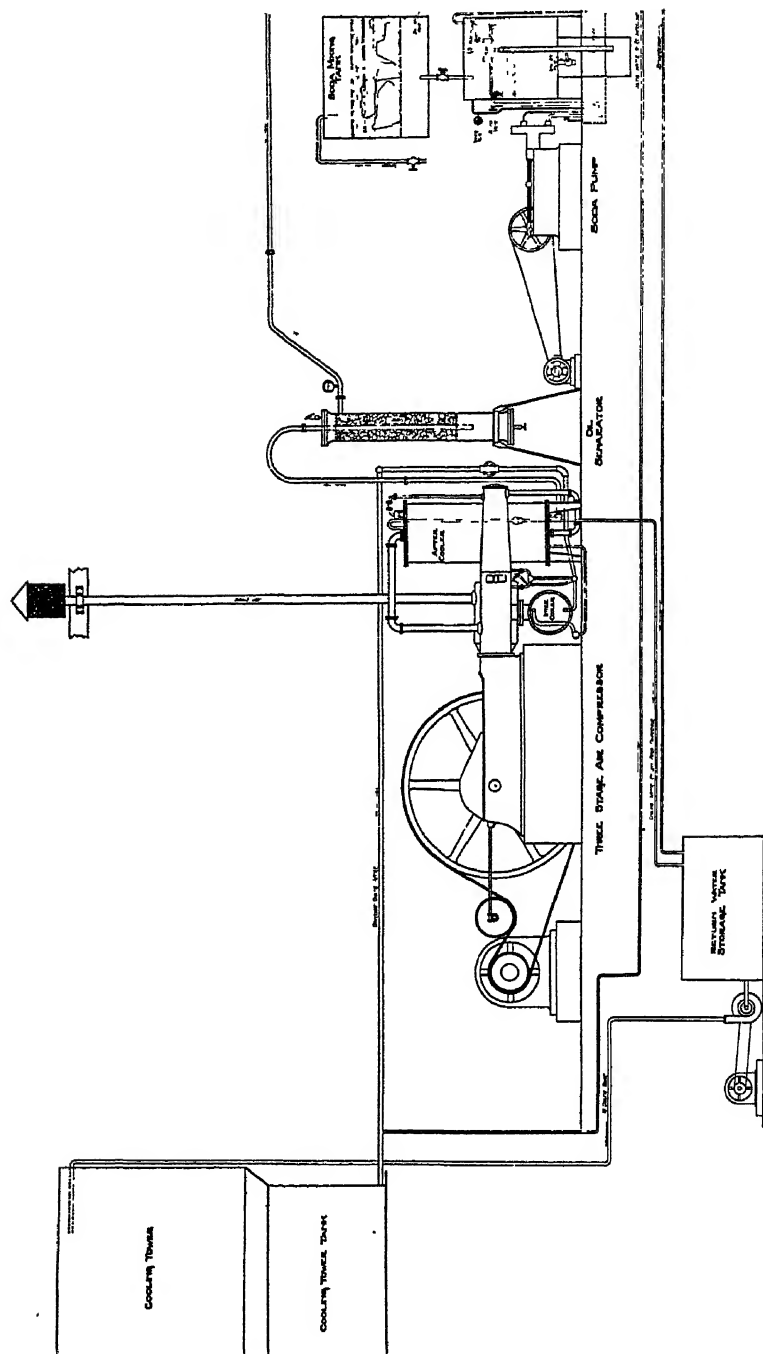


FIG. 1.—FLOW SHEET OF CLAUDE SYSTEM 75-LITER LIQUID-OXYGEN PLANT, CHUQUICAMATA.

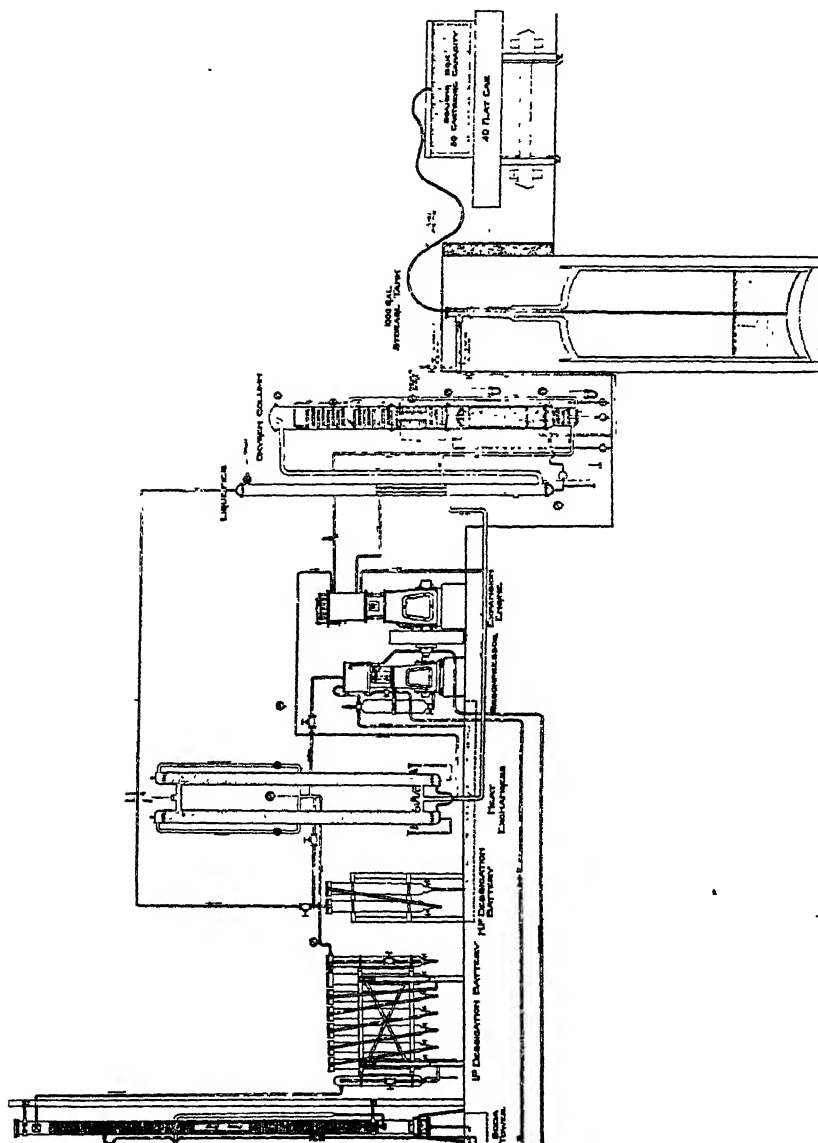


FIG. 1.—(Continued.)

2. The large amount of insulation material would tend to pack and lose efficiency.

3. Unless housed airtight it would collect moisture and lose efficiency.

4. Very long filling and discharge piping would be required, increasing evaporation losses.

Details of design of large double-walled vacuum-space containers were stated by Mr. Bedortha to be as follows:

1. It is essential that a metal be used which is impervious to gases (in so far as that is possible) and also which is not weakened by the low temperature of liquid oxygen.

2. All seams and joints must be vacuum-tight.

3. The walls of the vacuum space must be of such material that when highly polished they will exclude radiant heat.

4. The construction must be such that immediately after the polishing of the walls of the vacuum space is completed, the container can be completely assembled and the vacuum established. Thus it must be possible to quickly join the inner and outer neck tubes in a rigid, vacuum-tight joint.

5. The diameter and gage of the inner neck tube must be held to the minimum, allowing sufficient strength to support the weight and sufficient size to accommodate pipes for filling and discharging.

6. It is also necessary to provide a means of spacing the inner and outer shells which will not allow one shell to move with respect to the other during transportation and which may be removed when the container is in use.

7. Means must be provided for maintaining a vacuum corresponding to an absolute pressure of 0.0001 mm. of mercury.

8. Means must be provided to prevent injury to the container caused by the contraction of the metals at low temperatures.



FIG. 2.—ARRANGEMENT OF VACUUM PUMPING APPARATUS.

Three 1000-gal. vacuum wall containers were decided on to store the production of the 75-liter plant for a week. These containers were built by the American Welding Co. under the supervision of L. E. Bedortha, of the Purox Co., who tested them and found them vacuum-tight before shipment was made. The steel walls on either side of the vacuum space are $\frac{5}{8}$ in. thick and lined with sheet copper, polished to a mirror finish. The vacuum space was filled with dry nitrogen to prevent tarnishing of the copper during shipment.

One set of vacuum pumping apparatus is used for each container. It consists of a Gaede mercury diffusion pump, connected to the vacuum space through an air trap, and backed by a Cenco-Hyvac oil pump; the degree of vacuum is read on a Columbia gage. The arrangement of this apparatus, shown in Fig. 2, was worked out by Mr. Bedortha, and A. M. Rodgers, of the mine staff, set up the apparatus without difficulty and reëvacuated the containers to a degree closely approximating that estimated as possible when the containers were designed. Fig. 3 shows the average vacuum maintained while keeping the tanks constantly in operation, and shows the loss of liquid oxygen in liters per hour corresponding to various degrees of vacua.

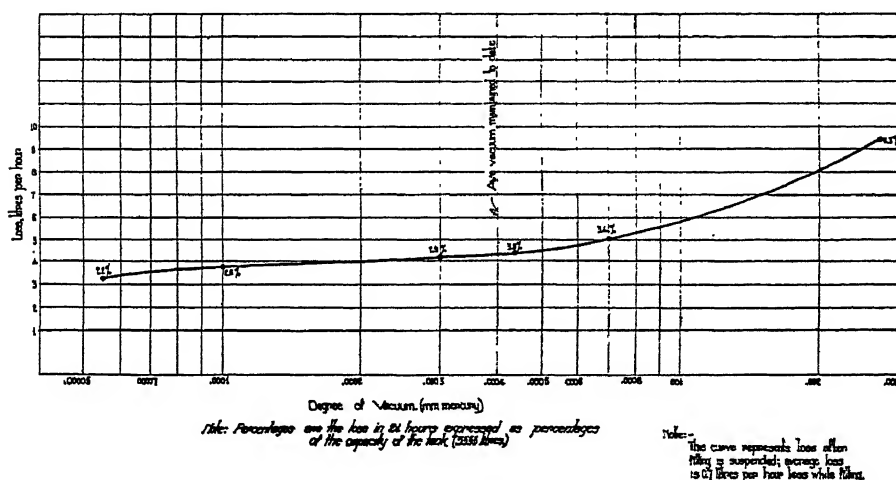


FIG. 3.—RELATION OF LOSS OF LIQUID OXYGEN TO DEGREE OF VACUUM IN 1000-GAL. VACUUM WALL CONTAINERS.

Pumping must be constant, at least during the time when tanks contain liquid oxygen to hold the necessary fine vacuum. It was not considered advisable to use a charcoal absorbent for small amounts of gas seeping into the vacuum space, as is done in smaller containers, because of the difficulty of its renewal.

The three containers have a total capacity of 10,000 liters and require one week to fill with liquid oxygen. The containers are set below floor level of the plant to allow them to be filled by a gravity flow from the liquid-oxygen draw-off valve.

FILLING AND DISCHARGING

The liquid oxygen is conducted from the draw-off valve to the storage containers in a $\frac{5}{8}$ -in. copper pipe inside a 3-in. hair-felt insulated housing pipe containing a counter flow of cold, evaporated oxygen gas.

The evaporation loss during filling is proportional to the length of filling pipe leading to the several containers. For the insulation above described this loss has been determined as approximately 0.2 liter per hour per lineal foot of filling pipe under average conditions of cold gas insulation.

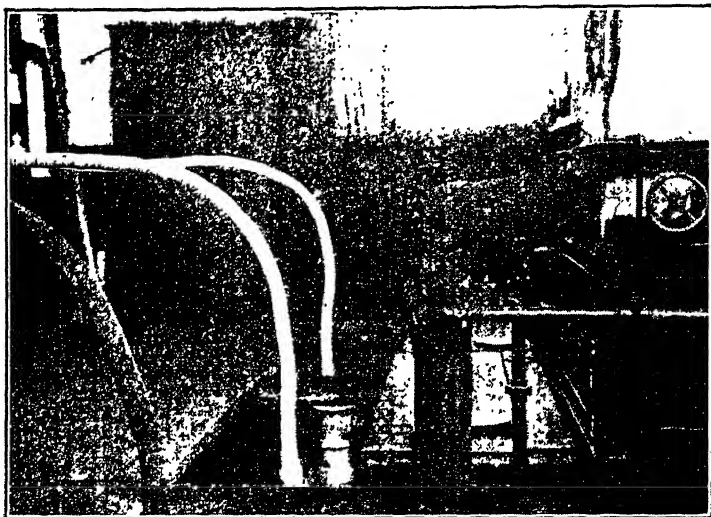


FIG. 4.—ARRANGEMENT FOR FILLING AND DISCHARGING LIQUID-OXYGEN STORAGE CONTAINERS.

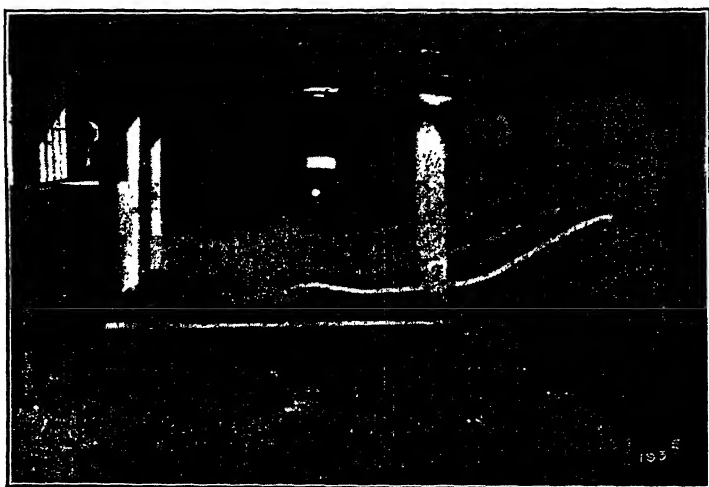


FIG. 5.—FILLING FOUR SOAKING BOXES SIMULTANEOUSLY.

The storage containers are discharged at 15 lb. per sq. in. air pressure through a cotton-covered $1\frac{1}{2}$ -in. flexible metal hose to an insulated manifold pipe, and from this pipe through $\frac{3}{4}$ -in. flexible metal hoses into four soaking boxes simultaneously. About 20 min. are required to fill

four soaking boxes in this fashion; one-half the soaking time of the larger cartridges.

MEASUREMENT OF PLANT PRODUCTION AND EVAPORATION LOSSES

Since the plant production is constantly varying slightly, and it is desirable to know both total production and the various evaporation losses with accuracy, it is necessary to keep records of the degree of vacuum in the containers, of the rise of the liquid during filling, of its fall when not filling, and of the gas evolved. The gas is measured in an aspirator under given conditions of temperature and pressure, and calculated to a corresponding loss in liters of liquid per hour. By making these various measurements under numerous different conditions, Mr. Rodgers has been able to state accurately the total plant production, the filling-pipe loss, the storage loss and the "cooling-down loss" when it is necessary to recool a container. On the basis of a week's production, filling three storage containers, the filling-pipe loss averages 5 per cent. of the liquid manufactured, and the total storage loss, including cooling-down losses, averages 22 per cent. of the liquid manufactured.

CARTRIDGE PLANT

This plant is in large part an adaptation of the pioneer work of George B. Holderer, of the Air Reduction Co. Mr. Holderer devised the mixture we are now using, consisting of 65 per cent. Bugbird (a coarse-grained carbon produced by a special process of burning wood pulp), and



FIG. 6.—SOAKING BOXES ON FLAT CAR AT L. O. X. PLANT.

35 per cent. Huber gas black. The coarse material gives the large cartridge mechanical strength, and the fine gas black largely determines its high rate of detonation, and supplies it with good absorptive and retentive qualities for the liquid oxygen. The wrapper or bag material used is No. 8 cotton duck. Two sizes of cartridges are being made, one 7 in. dia.

by 20 in. long, weighing 8 lb., for column loading in 8-in. drill holes, and one $5\frac{1}{4}$ in. dia. by 12 in. long, weighing $3\frac{1}{2}$ lb., for loading in 6-in. chambered drill holes.

The carbon material is shipped from the United States in bags. It is mixed, in the proportions given above, in an electrically driven batch-mixing machine and elevated with a bucket conveyor to the hopper of the packing machine, as shown in Fig. 7. The packing machine² (Fig. 8) was designed to be heavier and larger than those in use in the United States

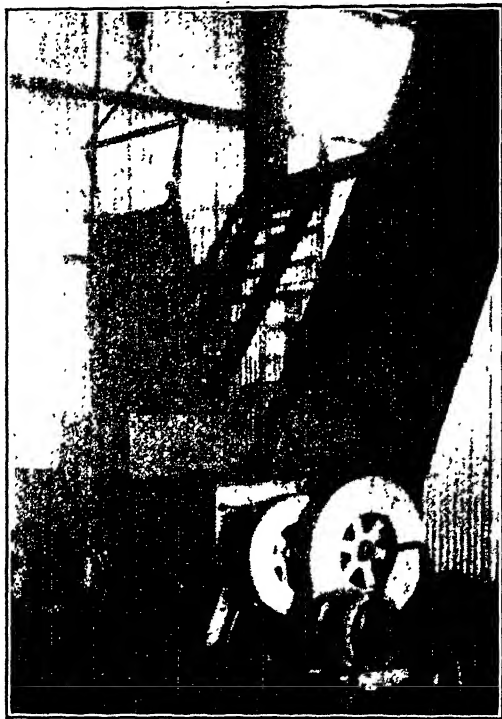


FIG. 7.—BATCH MIXER AND ELEVATOR FOR CARBON BLACKS.

to pack larger cartridges than have been used elsewhere. As many as 1000 cartridges each weighing $3\frac{1}{2}$ lb. have been packed in an 8-hr. shift by three Chilean workmen. This includes mixing the carbon but does not include time of cutting and sewing bags, which is a somewhat slower operation.

SOAKING OF CARTRIDGES

The soaking boxes are 36 in. wide by 72 in. long by 22 in. high, inside measurements. Each box holds 50 of the 7 by 20-in. cartridges or 125 of

² F. W. O'Neil and H. Van Fleet: *Op. cit.*, 695.

the 5½ by 12-in. cartridges. Twelve of these boxes are equivalent to the 10,000-liter storage capacity. The soaking boxes are carried on two 40-ft. flat cars, which have steel roofs to shade the boxes from the sun.

Two kinds of boxes have been built, some with 1-in. hair-felt insulation and others with 4-in. balsa wood insulation. The latter have proved nearly twice as efficient as the former in keeping down evaporation losses.

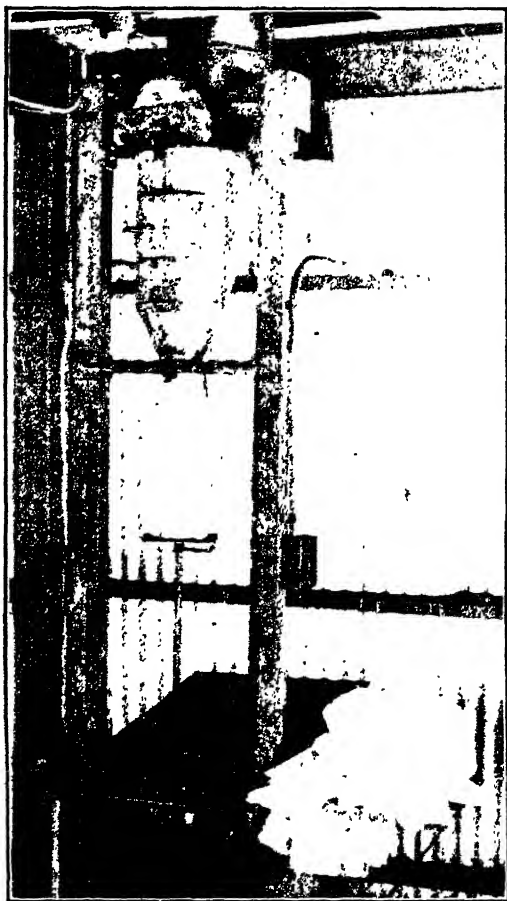


FIG. 8.—CARTRIDGE-PACKING MACHINE.

The evaporation curve (2), shown in Fig. 9, indicates that the 7 by 20-in. 8-lb. cartridge will absorb 3.4 times its own weight of liquid oxygen but this is true only when the cartridge is kept nearly fully immersed in liquid oxygen throughout the absorption period of 40 min. In operating practice it has not proved economical to soak this cartridge with more than 2.9 times its dry weight of liquid. When more than this amount of liquid

is used the cost of the dirty waste liquid remaining in the soaking boxes after removal of the cartridges exceeds the economic gain from added explosive strength which is obtained by the use of this greater amount of liquid. On curve 2, Fig. 9, it can be seen that starting with a 2.9 ratio of oxygen to carbon is the equivalent of a 15 min. lapse of time in the bore-hole, from a 3.4 ratio.

For a shot fired in $1\frac{1}{4}$ hr. after starting to remove the cartridges from the hair-felt insulated soaking boxes, using $5\frac{1}{4}$ by 12-in. cartridges, of the total liquid poured into the soaking boxes 41.8 per cent. was usefully

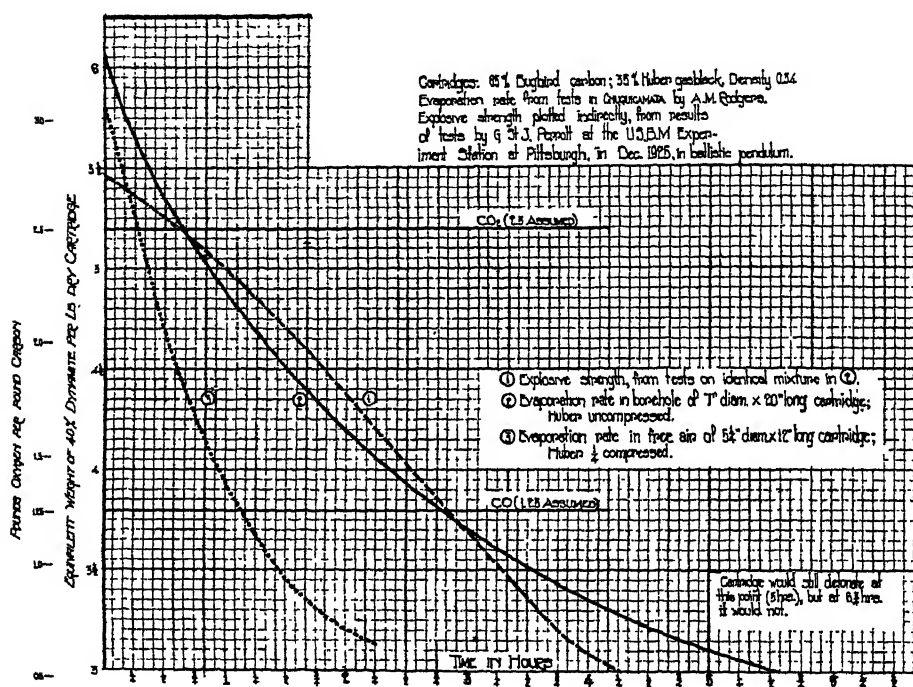


FIG. 9.—EVAPORATION RATE AND EXPLOSIVE STRENGTH OF CARTRIDGES.

employed at the time of explosion, 64.6 per cent. was absorbed in the cartridges at the time of their removal from the soaking boxes, 23 per cent. represented evaporation loss during soaking and transport, and 12.4 per cent. was dirty waste liquid remaining in the boxes after removal of cartridges. Enough of this dirty liquid is filtered through canvas to use around the liquid-air traps of the vacuum apparatus.

Table 1 shows the complete distribution of the liquid oxygen for a blast under the above conditions, as percentages of the liquid manufactured. Smaller losses than those shown in the table have been obtained when using the large 7 by 20-in. 8-lb. cartridges and the balsawood insulated soaking boxes.

TABLE 1.—*Distribution of Liquid Oxygen for One Blast*

	PERCENTAGE OF LIQUID MANUFACTURED
<i>Filling pipe loss:</i> (3.97 l. per hr. ave.).....	5.1
<i>Storage loss:</i>	
Storage while filling (48 hr.—3.65 l. per hr.).....	4.7
Storage while not filling (85 hr.—4.2 l. per hr.) using three full tanks.....	9.5
Cooling down tank.....	5.2
Remaining in tank because of convex bottom.....	3.2
<i>Total storage loss</i>	22.6
Discharge loss.....	
Loss, cooling boxes (hair-felt insulated).....	16.7
Loss, cooling cartridges.....	
Evaporation loss during soaking and transportation (approx. 2-hr. period).....	
Loss from cartridges during loading, testing and firing.....	16.5
Liquid in cartridges at time of firing—(oxygen-carbon ratio—1.88)...	30.2
Liquid in cartridges at time of removal from soaking boxes.....	46.7
Liquid recovered for thermos bottles (for vacuum pumps).....	2.0
Liquid remaining in soaking boxes—wasted.....	6.9
<i>Total</i>	100.00

BLASTING PRACTICE

All of the L. O. X. used to date has replaced a free-running 65 per cent. strength ammonia dynamite in churn-drill holes. No change in the spacing of the drill holes or in the amount of springing done in those holes, which are partly pocket-loaded, has as yet been brought about by the use of L. O. X. This is because of our large cartridges and average time of firing a blast—1 hr.—the decreased density of the L. O. X., as compared to the free-running dynamite it replaces, is slight, and is more than offset by its increased speed as compared to the dynamite. Tests made by the U. S. Bureau of Mines on our cartridge mixture show it to have a rate of detonation of 5200 m. p. s., which is 40 per cent. greater than that of the ammonia dynamite it replaces, though its propulsive effect is not so great.

The locations of all drill holes are staked for the churn drills by the mine engineers, who later survey the exact locations of the drill holes and the position of the bench face. With these data, shot plans are made up, the explosive charges calculated in the mine engineering office, and given to the powdermen. Constant coöperation between the powdermen and the engineers, and study of the results of all shots, together with any deviation in the powderman's loading sheet from that of the engineers, keeps the loading factors revised up to date according to latest experience in all sections of the mine. It has been found that the L. O. X. can be loaded, with good results, about 20 per cent. lighter, considering its weight

at time of firing, than the 65 per cent. ammonia dynamite used in a corresponding location. Knowing in advance the approximate time it will take to load and shoot a given shot, the weight of the L. O. X. at time of firing can be determined from the curves 2 and 3 shown in Fig. 9.

No attempt has yet been made to use L. O. X. for springing or secondary blasting, as for this work the relative amount of dynamite replaced

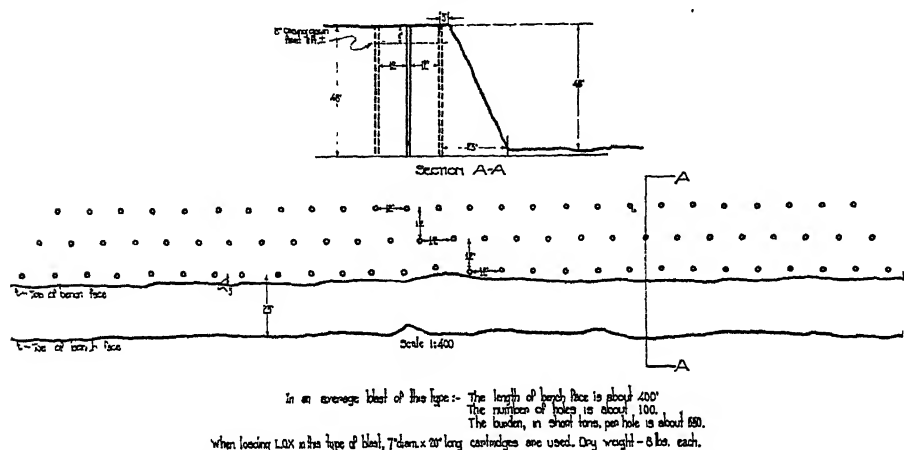


FIG. 10.—TYPICAL CHURN-DRILL HOLE SPACING FOR COLUMN LOADING.

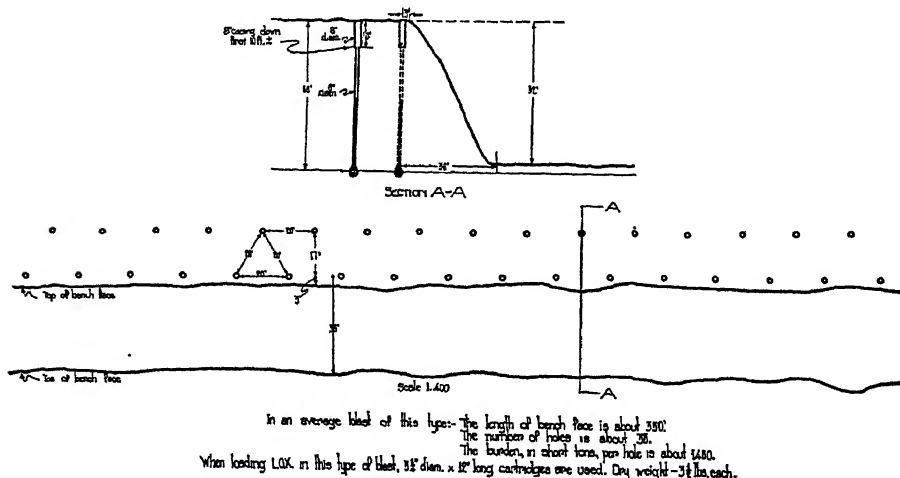


FIG. 11.—TYPICAL CHURN-DRILL HOLE SPACING FOR POCKET AND BARREL LOADING.

would be small and transportation and storage costs of the liquid oxygen would be high.

No attempt is made to load very ragged or crooked holes with L. O. X. Such holes form a small percentage of the total drilled and are segregated by the powderman after looking over all the holes with a mirror before the loading sheet is made up. These holes are loaded in advance with dynamite.



FIG. 12.—UNLOADING SOAKED CARTRIDGES AT BLAST.



FIG. 13.—BEFORE TYPICAL L. O. X. BLAST.

FIG. 14.—AFTER TYPICAL L. O. X. BLAST.

All cordeau used in the holes is wire-wound and is placed in advance of the arrival of the L. O. X. train. Tamping of the holes and surface connections of the cordeau follow at some distance behind the actual charging of the holes with cartridges.

Fig. 12 shows the cartridges being carried in a tray by two men down a bridge from the soaking boxes to the shot. The average running time of the L. O. X. train from the liquid-oxygen plant to the shots on various benches is 20 minutes.

Costs

During the month of October, 1927, a total of 6579 cartridges weighing $3\frac{1}{2}$ lb.—total dry weight, 23,027 lb.—were manufactured at the cost shown in Table 2.

TABLE 2.—*Cost of Cartridges for October, 1927*

	Total Cost	Cost per Pound of Cartridge
14,968 lb. Bugbird carbon black.....	\$1600.08	\$0.069
8,059 lb. Huber gas black.....	1195.15	0.051
2,192 yd. of 36-in. 8-oz. duck.....	544.08	0.024
Brass clips and thread.....	85.54	0.004
Labor.....	83.98	0.004
Supervision and overhead.....	69.08	0.003
Power and maintenance.....	20.85	0.001
	<u>\$3598.76</u>	<u>\$0.156</u>

During October, 126,000 lb. of liquid was manufactured at the cost given in Table 3.

TABLE 3.—*Cost of Liquid Manufactured in October, 1927*

	Total Cost	Cost per Pound of Liquid
105,526 kw.-hr. power at \$0.00868.....	\$ 915.97	\$0.007
Labor, ten men including vacuum-pump operators.....	543.60	0.004
Supervision.....	370.00	0.003
Maintenance.....	240.00	0.002
Chemicals.....	94.14	0.001
Miscellaneous (water, lubricants, etc.).....	47.13	0.001
(Interest and amortization on investment, $(10\frac{1}{3}\%$ yearly; \$15,450 per year).....	1290.00	0.010
	<u>\$3500.84</u>	<u>\$0.028</u>
Cost per liter.....		<u>\$0.0716</u>

Table 4 shows results for the five-month period June to October, 1927, inclusive.

TABLE 4.—*Results during Five Months*

Tons broken.....	937,195
Pounds liquid manufactured.....	494,295
Pounds liquid effective at time of firing.....	163,730
Pounds of dry cartridges used.....	83,830
Pounds of L. O. X. effective at time of firing.....	247,560
Tons broken per pound L. O. X. at time of firing.....	3.78
Pounds liquid manufactured per pound of dry cartridge.....	5.9
Pounds dynamite replaced in breaking 937,195 tons.....	312,000
One pound cartridge at 0.156.....	\$0.156
5.9 pounds liquid manufactured at 0.028 per pound.....	0.165
<hr/>	
3.0 pounds L. O. X. effective at time of firing.....	\$0.321
1.0 pound L. O. X. effective at time of firing.....	\$0.107
247,560	
312,000 \times 0.107 = \$0.085 cost of L. O. X. replacing 1 lb. of dynamite at \$0.16.	

CONCLUSIONS

The 5.9 lb. liquid manufactured per pound of cartridge is higher than the 4.5 ratio obtainable where daily shooting with small container storage is in effect. In consequence of this high ratio, costs are higher than those originally thought possible of achievement. Nevertheless distinct savings are being made by the use of this explosive and the blasting results with this explosive in the harder rock at the mine have been entirely satisfactory.

DISCUSSION

G. B. HOLDERER, New York, N. Y.—We have had no serious difficulty at any time. Carbon monoxide does exist, but we have never had enough to bother a man.

J. M. RIDDELL, Wolfpit, Ky.—Does the unit cost include the plant depreciation?

G. B. HOLDERER.—The authors do not seem to have figured in depreciation and overhead. At Enos, the figure of 6 c. is an operating cost. Overhead will add another cent, so that is really 7 c. The 7 c. would include everything. With the second unit, which will go into service early next month, that will be brought down to 5 and 5 ½ c. I believe they have a system of bookkeeping that takes into consideration depreciation and overhead on each item as they go along.

A. O. IHLSENG, New York, N. Y.—The cost per cartridge was 58 c., which is on the basis of about 4 ½ c. per pound of cartridge. Recently, I have had more or less to do with strip coal mining. I have just returned from a two months' trip covering some of the largest strip mines in Illinois and Indiana. The present problem of strip mines is one of handling the rock overburden. When strip mining first started, the "easy picking" was taken with a small shovel. As time progressed it has been necessary to take a heavier overburden, which necessitated the handling of considerable rock. Now it is absolutely essential to consider 50 or 60 ft. of overburden to get any area of coal in which such an expensive outfit as stripping shovels can be used and pay for themselves.

Of the various properties that I have examined in the last two months, two are operating with liquid oxygen. Both have heavy rock overburden. The Enos company, as Mr. Holderer says, runs about 35 ft. of rock on a 40-ft. overburden. Most of that is sandstone and limestone that is very tough. Their experience in handling this overburden with ordinary dynamite or black powder was that the shovel was in constant difficulty because of very large boulders. Much of that rock had to be pop-shot to enable the shovel to pick it up. Since liquid oxygen has been used, the mass has been thoroughly shattered, and the shovel can handle the overburden with ease.

Two years ago I examined a stripping operation in Southern Illinois. They had been using black powder and had allowed the pits to get into a frightful condition, so that the mine was practically profitless. The rock overburden was in great masses and entirely blocked the pit. Later they put in a liquid-oxygen plant, and when I examined the same property during the last two months I found the mine in an entirely different condition and operating at a profit. The overburden was thoroughly shattered, and there was no difficulty whatever in handling it with a steam shovel. From a mine that threatened disaster has grown one that is doing very well. In two different Indiana properties close together, the Enos and a property about 10 miles north, the cost per ton of coal is about the same. One is using black powder and the other is using liquid oxygen, but the property 10 miles north has a rock which can easily be shattered with liquid oxygen at a cost of at least 15 c. a ton less. There is no question about that, because there is very little limestone or sandstone. It is practically all shale, which is readily breakable.

I discussed with Mr. Holderer once the question of shooting larger unit areas. I make the claim that the economic use of liquid oxygen can be still further improved over the practice of shooting 20-ft. squares, that is, with holes 80 ft. apart. With the ordinary coal seam of 5 ft., an area of that kind will uncover about 80 tons of coal. With the high shattering power of liquid oxygen, I am confident that 30-ft. squares with holes 30 ft. back from the face, uncovering 210 tons, will reduce the cost of shooting overburdens per ton just about 50 per cent. That is quite material, since the margin of profit in coal is very small and every 10 c. counts.

Ordinarily, a cartridge is used for every 6 ft. of hole. I think by the utilization of about 50 per cent. more liquid oxygen in this 30 ft. spread of holes, better results per ton of coal can easily be obtained.

S. J. JENNINGS, New York, N. Y.—Liquid oxygen was used at the Real del Monte mine, Mexico, for several years.³ The net result as far as we were concerned was that, at the price of dynamite that we could then obtain, liquid oxygen was very much cheaper than dynamite; but as soon as the dynamite factory realized that it was being driven out of business, it reduced the price, so that it was cheaper for us to use dynamite than to use liquid oxygen. So we reversed ourselves and are now using dynamite.

We had one mine that used nothing but liquid oxygen; a mine that was producing somewhere around 500 tons a day. We were contemplating putting in liquid oxygen for the whole production of some 3000 tons a day when the dynamite company reduced the price of its product.

T. MARVIN, Wilmington, Del.—This discussion indicates one use for the compilation of facts contained in the handbook on Drilling and Blasting in Some American Metal Mines. I refer particularly to the discussion of tamping and stemming of holes, in which it was said that very few of the metal mines are using tamping. I

³ See M. H. Kuryla and G. H. Clevenger: Liquid-oxygen Explosives at Pachuca. *Trans.* (1923) 69, 271.

refer you to any of the pages in the front of the compilation. On page 11, for instance, under the division of shaft sinking at copper mines, mention is made that the Moctezuma Copper Co. uses mill slimes for tamping; the Nevada Consolidated Copper Co. employs none; the next, the Old Dominion Co., uses damp, sandy soil, and so on through the compilation. In other words, there is the testimony of nearly 100 mine managers or superintendents concerning stemming.

Questionnaires were sent out from Wilmington in October, 1927. The data were returned splendidly and quickly. I believe the first answer came in on November 1; therefore the material is not more than four months removed from its source.

It would be impossible to make the large number of sketches, used in the compilation, on true scale, so I had to standardize on a certain size for shafts, a definite diagram dimension for drifts and so forth.

The compilation really represents the present "how" of drilling and blasting. It was felt that men in mining today, or those contemplating opening up new properties, are interested somewhat in the practical phase. All of these data were submitted by men on the job.

I know there may be some criticism that the number of tons of rock broken by each pound of explosive should have been included. Other factors that might be added are the drilling speed of the drill, the feet per minute, and the hardness of rock, but there had to be a line drawn somewhere and the tabular headings illustrate that line.

Personally, I felt that the type of explosive used, naming the type of rock, and giving the exact number of cartridges used in every hole, practically answered the question for the man who is seeking definite information on how to blast a certain rock or ore if accompanied by some certain procedure data. I am much interested to know if the different phases and factors covered in the compilation satisfy, or if there are additional conditions which men at the mines would really like to possess.

I might go through a typical example of the use of the compilation. Assume, for instance, that a man is interested in shaft sinking in quartzite. Turn to page 196, under the first index of shaft, "Type of Ore or Rock Blasted." Go through the list of rocks. Under each are listed certain numbers. These numbers refer to the mine numbers; that is, the numbers of the mines included in the compilation. For instance, Mine No. 9, which is the first one under quartzite. Turning to page 192, you will find that No. 9 happens to be a copper company. The first column refers to the shaft data page, which is page 8, and on page 8 is the drilling and blasting information concerned with shaft sinking at that property in quartzite.

Also, on page 85, which is in the shaft section, you will find the shaft diagram used at this mine for drilling quartzite. There are the dimensions of the shaft, exactly how the holes are placed, how deep and how many cartridges in each hole. If you wish to know how they use stemming, refer to the data following on the same page, 8 or 9. By cross index and reference table, one interested in blasting in any type of rock included within the borders of the compilation, for any purpose, can find that material listed and readily available.

S. J. KIDDER, Eustis, Que.—Were any data collected as to the preparation of the stemming and the method by which it is delivered to the mine?

T. MARVIN.—Some of the men who sent in data referred to that on the questionnaire. I believe most of those who mentioned the subject said that the stemming material was delivered to the mine and was used in cartridges. That is, empty cartridges had been filled either at the shaft bottom or at the surface, and were delivered to the miner filled, so that all the miner had to do was to place the material or the stemming cartridges directly in the hole.

S. J. KIDDER.—I think the preparation of the cartridge for the miner is a determining factor with a great many operators in regard to the use of stemming. A number of years ago, in New Mexico, we found it hard to induce the miners to use stemming unless filled cartridges were supplied. Then the question of filling the cartridge came up and we found that stemming with 20 per cent. moisture was pretty hard to put into the cartridge, so we designed a cylinder about 10 in. dia. with a screw feed. It had a stand pipe of the same diameter; had a head motion, and three spouts on the discharge, over which we could slip the empty cartridges. The operator would then start the machine, and fill three cartridges at once. For a mine producing only 5000 to 6000 tons a month, it was a satisfactory arrangement. In larger operations, the preparation of the stemming ought to be taken up so that it could be handled most economically. I wonder if any of the mines have gone into that thoroughly?

C. S. HURTER, Wilmington, Del.—Within the last 60 days I have made a study of the holding power of several different kinds of tamping. The results showed that wet sand was the best; properly moistened clay, second; dry sand, third; dry clay, fourth; and slack coal the worst of all.

D. HARRINGTON, Washington, D. C.—There is considerable delusion in this matter of stemming. I have gone into mine after mine, metal mines, where they are supposed to use non-combustible stemming. If the miner has it at his finger tips he may use it; very frequently he does not use it. Too frequently, he does not have any place convenient to secure it. In coal mining, it is an axiom that it is dangerous to the miner himself as well as to anybody in the mine to use coal dust. I have made examination after examination of mines where there is a strict rule that no coal dust shall be used for stemming, and in more than half of the instances the miner will use coal dust and nothing else. If the clay stemming is right at hand he will generally use it; if he has to go 300 or 400 ft. to get it, he will use the coal dust. Many of the mines I have seen prefer to use stemming; they try to bring the stuff to the miner, but very frequently it does not get to him.

R. K. WARNER, New Haven, Conn.—I am much interested in Mr. Marvin's compilation. A year ago I collected similar data for Peele's Handbook. This was compiled from a questionnaire similar to the one Mr. Marvin used. After tabulating results from about 103 late examples I was able with somewhat indifferent success to derive figures for the powder consumption, feet of drill hole, etc., that might be expected under normal conditions in different types of rock. Probably with the additional data that Mr. Marvin collected this can be carried further.

In connection with stemming, I would say that I have done a considerable amount of traveling within the last two years among the mines and have found that the miners will in general use stemming only when it is bagged and prepared for them and issued with the powder.

R. R. VAN VALKENBURGH, Wharton, N. J.—The psychology of the thing appears to be one of the most important points. For instance, in regard to stemming, I have found that if the miner is working on an ordinary day's pay scheme he is very apt to overlook the stemming, whether or not it is furnished to him at the face, but if he can be placed in a position to make a bonus on saving in explosives, he begins to ask for stemming, and in our case, to ask for spacing blocks.

We find that the contractors, the men who are driving development operations on a piece-work basis, want stemming and want spacing blocks. It proves to me that whether it is psychological or practical, the stemming should be all right.

I prefer the clay whether it is wrapped in cartridges or not. However, in our mine in New Jersey we use fine ore; anything we can scrape up on the ground. The miner carries a number of stemming bags in his knapsack and fills them himself.

T. MARVIN.—You will notice in the shaft section of the compilation that nearly every mine, or the majority of mines, use stemming. I believe it is true also that shafts are driven by contract more than the drifts and the crosscuts.

On page 33—I just happened to open that page—under “Drifts,” out of eight mines in drifts only one uses stemming. One out of eight uses stemming in drifts, which may indicate that the use of stemming is the practical measure of decreasing blasting cost used by contractors who are interested in doing their work as cheaply and as efficiently as possible.

B. F. TILSON, Franklin, N. J.—I have often wondered whether the virtue of stemming was not due chiefly to the inertia of a mass. At that rate, gelatin dynamite is rapid enough so there is not the material wedging effect that is often pictured in making use of the stemming. Water hammer is a force that reaches serious proportions dependent on inertia, and, similarly, the deeper the stemming that is used, the greater the mass, and the more effective it is, regardless of the mechanical wedging action.

I would like to raise the question for discussion as to whether inertia is one of the factors. If so, I would differ with what Mr. Van Valkenburgh has suggested. His preference was for clay. I would say that he had an ideal stemming because of its great density.

We have used clay. Since we have had a similar, but heavier ore than his, in the Franklin mine, we have used the ore in paper cartridges for stemming. He referred to the spacing sticks, and again I think the explosive experts will say that we are throwing away a lot of explosive when spacing the cartridges. It has not seemed to work out that way with us. We seem to be able to economize in explosives and produce the results by spacing the cartridges, and the fact that the men themselves have for years followed that practice indicates that the results have been satisfactory to them, whether it was on company account or contract work.

Theoretically, explosive experts tell us how the rate of detonation is lessened by traversing through poor material such as wood, or through air; that the efficiency of the explosive is decreased. Undoubtedly, those things are theoretically correct, but sometimes economic factors do not balance with pure theory.

The spacing of the dynamite by wooden spacers may gain more than the loss of efficiency in blasting. We have felt time and again—possibly we are wrong—that the wide, more or less uniform distribution of explosives in our ground at any rate would be advantageous and would produce the most effective results. In other words, holes drilled with the explosive spaced over a long distance in the hole produced the most effective results in the breaking of ground to a fine condition, which lessened the amount of handling and sledging. We believe money can be saved by the better distribution of the explosive. Regardless of laboratory theory, the dollar sign comes in when balancing out the total operation.

I remember we had an experience some years ago in our open-cut work, in which we were blasting a limestone for filling purposes. There were limestone quarries near us in which they were breaking limestone for a blast furnace. There they were using well-drill holes, big holes. Of course, the development of the hammer drill permitted placing the holes closer together in the bench and putting in a lot of relatively small holes to blast down the limestone. The man who was at that time in charge of the churn drilling at this property later happened to associate himself with one of the explosives companies and he was very keen to come and prove to us in both respects how foolish we were. He later left that job and took a job as manager of some other rock quarries, and we were very much interested to find that instead of using a churn drill he had adopted our quarry practice of the hammer drill.

Instead of using tamping sticks and cutting them for spacers between the cartridges the men very frequently take ordinary pieces of powder boards, split off slivers, and use them as spacers. I think that the detonation goes through the air space without any question. We do not have any trouble with the transmission of the detonation. We do not have so-called stinkers, in which the powder has been ignited and is burning. It seems possible to get a useful effect from gelatin dynamite by abusing it very considerably theoretically.

C. S. HURTER.—In connection with that spacing method, I have been in places where it is used with excellent results and can be considered the best possible practice. It is particularly effective where deep holes are drilled to break off thin slabs, such as in widening drifts, etc. I do not believe that the explosion is carried through the space, which is known among scientists as explosion by influence. To my mind the explosion of the different cartridges is due to the explosion of one cartridge driving the stick of wood violently into the next, and so on through the different spaced charges. An investigation of this practice of blasting has brought out the following points: (1) The stick of wood should be small enough so that there is no possibility of it binding in the bore hole; (2) the ends must be square and in actual contact against the ends of the cartridges so that it cannot slip by them; (3) hard wood is more efficacious than soft wood where this practice is employed. For breaking off long thin slabs I consider this one of the best practices in existence.

G. ST. J. PERROTT (written discussion).—The operation described by Schultz and Hunter marks a distinct advance in the application of L. O. X. to open-cut blasting. In particular, the development of the 1000-gal. storage container enlarges the scope of L. O. X. in open-cut blasting because it permits accumulation of sufficient quantities of liquid oxygen for large blasts at less frequent intervals and with a plant of smaller capacity than was necessary when the total production of the plant had to be used daily.

It is interesting to compare the comparative propulsive strength of L. O. X. and dynamite as determined by the ballistic pendulum and as shown in actual blasting at Chuquicamata. From the figures obtained at the Explosives Testing Station of the U. S. Bureau of Mines at Bruceton, Pa., 1 lb. of dry L. O. X. cartridge (oxygen-free) is equivalent in propulsive strength to 4.5 lb. of 40 per cent. ammonia dynamite, if fired at an oxygen-carbon ratio of 1.88. This would be the equivalent of about 4.0 lb. of 65 per cent. ammonia dynamite. From the data in Table 4, calculation indicates that 1 lb. of dry L. O. X. cartridge replaced 3.65 lb. of 65 per cent. ammonia dynamite in actual blasting. This shows a close approach to the value obtained by means of the ballistic pendulum.

Report of Committee on Metal Mine Ventilation

PRESENTED BY D. HARRINGTON, CHAIRMAN

(New York Meeting, February, 1928)

THERE has been, during the past year, a wealth of report data and discussion relating directly or indirectly to various phases of ventilation of metal mines, and many of the data are from foreign sources. The phase of the subject most generously accorded the attention of the technical press is that of cooling mine air, with much less attention to other fairly intimately associated features such as fan performance, friction factors, fires, mine gases, mine lighting, health of miners, and ventilation standards.

COOLING MINE AIR

South Africa, with its very deep mining on the Rand and with fairly high working temperatures, contributes the most definite data on the subject of cooling mine air, and undoubtedly the most interesting issued are those from a paper by J. H. Dobson,¹ in which he gives a comprehensive review of the subject as gleaned largely by analyzing the data in three papers.²

Dr. Dobson's summation is decidedly interesting. In general it is recognized that the introducing into the mine and circulating to the working places of large volumes of pure air is the most effective method of giving deep, hot metal-mine workings reasonably efficient working conditions. A tabulation as to the cost of securing various amounts of cooling by circulating various volumes, from 230,000 to 760,000 cu. ft. per min., indicates that there are cost limits beyond which increase of quantity of air in circulation is impracticable, as the annual cost of circulating 230,000 cu. ft. of air per min. in a certain mine at a depth of

¹ J. H. Dobson: The Engineering Aspects of Mine Cooling. *Jnl. South African Inst. of Mining Engrs.* Abstracted in *South African Min. & Engng. Jnl.* (1927).

² A. J. Orenstein and H. J. Ireland: A Contribution to the Study of the Influence of Mine Air Atmosphere Conditions on Fatigue. *Jnl. South African Inst. of Engrs.* (1921) 19, 126, 186.

M. O. Tillard and E. C. Ranson: Rock and Air Temperatures in Deep-level Mines. *Jnl. Chem. Met. and Min. Soc. of South Africa* (1926) 26, 184.

A. Mavrogordato and H. Pirow: Deep-level Mining and High Temperatures. *Jnl. South African Inst. of Engrs.* (1927) 25, 101.

7500 ft. would be about \$36,000 while the annual cost of circulating 760,-000 cu. ft. per min. would be about \$480,000.

It is brought out that refrigeration of air at the surface is expensive and relatively ineffective; that "wholesale" cooling methods by compressed air are expensive; liquid-air cooling is even more expensive than compressed-air methods; and an attempt to cool mines by air drying by silica gel, calcium carbide and similar means is ineffective. Local use of compressed air blowers, as at working faces, is held to be helpful; circulation of cold water piped from the surface to deep workings was held to have possibilities towards cooling of mine air; the use of small-unit portable refrigerating plants to be located near working places was given a partial tentative approval; and the use of ice locally in small ventilating currents appears to have the greatest possibilities where ordinary methods are ineffective.

The conclusions of Messrs. Tillard and Ranson³ are as follows:

"(a) A ventilating plant of maximum capacity should be provided.

"(b) The strictest attention should be paid to air control so as to obtain the maximum benefit from the air at one's disposal.

"The control and concentration of air in the lower levels of the mine is of the utmost importance.

"(c) Much benefit can be obtained as far as comfortable conditions of working places are concerned, by creating and maintaining separate ventilation districts, by which means one is able to have some control on heat acquired by the air traversing the mine.

"(d) General cooling of the mine air by means of a central refrigerating plant on the surface or underground is not hopeful.

"If the system of general cooling of mine air is to be a success, we feel that it will be by means of small portable units, if the difficulty of getting rid of the condenser heat can be overcome.

"(e) Conditions in dead ends can be very much improved by the use of ice as a means of locally cooling the ventilating air; in fact, we might say that the problem of heat in these working places can be solved by these means.

"(f) Much benefit can be derived from compressed air boosters for locally cooling working gangs in places where the air movement is slight.

"(g) To generally cool the mine by means of cooling the mine water supply is a doubtful proposition.

"(h) No doubt benefits would accrue from insulating the air ducts or airways of the mine, but would probably be found difficult of application on these fields.

"(i) Very material benefits would be gained if the use of water underground could be cut down, so as to increase the evaporative power of the air.

"(j) There appears to be little hope of success working on the lines of air drying or dehydration by chemical means.

"(k) From a physiological point of view acclimatisation of the workman by arranging a suitable probationary system is a most important factor."

In a short discussion of the Tillard-Ranson paper, R. H. Flugge-deSmidt gives some interesting data as to handling high temperatures in

a coal mine in the Ruhr district of Germany, where, at depths between 2600 and 3250 ft., rock temperatures up to 111° F. are encountered and the mining regulations are that if the air temperature is above 82.5° F. the working period is 5 hr. instead of the usual 6 hr. Air volume was increased from 350,000 to 700,000 cu. ft. per min. and while the percentage of short or 5-hr. shifts had been above 80 in June, before increasing the air volume, this percentage dropped to zero by February, rising to 25 per cent. in the following summer. In order to make a further reduction in underground air temperatures, a cold-water "radiator" was installed in a main intake air crosscut 3150 ft. below the surface, the radiator being composed of a large number of 2-in., 3-in., and 5-in. pipes of considerable length through which cold surface water was circulated. The system was not wholly successful until an ammonia refrigerating plant was placed at the intake shaft collar, which cooled the water from about 60° to about 34° F. The plant operating with 22 cu. ft. of water per min., lowered the temperature of 250,000 cu. ft. of air from 73° F. on the surface to 68° F. at a point 1½ miles underground, the cost of running the plant being \$110 per shift; it is stated that the increased output averaged about \$172 per shift and that the cost of the plant, approximately \$17,500, was saved in about two years.

In a discussion of the Tillard-Ranson paper⁴ it is stated that in using 100 to 250 lb. of ice in a ventilating pipe delivering 1500 cu. ft. of air per min., it is possible to maintain an air temperature at a hot face 11° to 12° cooler than the rock temperature, and one instance is given where by this ice method the air at the face was reduced from 89.4° F. to 73° F. dry bulb. The ice is spread over the last 10 or 12 ft. of the ventilating tube; no specific figures are given as to ice consumption, but the ice method appears to have the best possibility for local air cooling when air movement alone becomes insufficient. It would appear, however, to be feasible to give ample cooling by air movement alone when dry bulb temperature is below 90° F.

Probably the most unusual feature of the more recent South African discussions on the cooling of mine air is the tendency to advocate methods that will hold air humidity to a minimum and to recommend the abandonment or at least the lessening of the use of water in drilling and in the watering of muck piles, face regions, etc., methods which have been developed largely to combat the dust hazard with miners' consumption in the background.

MINE FIRES .

In metal mining, ventilation is so closely linked with mine fires that the two are practically inseparable. While control of air currents by mechanical ventilation is decidedly desirable at time of fire in a metal

⁴ *Queensland Govt. Min. Jnl.* (1927) 28, 473.

mine, the three metal-mine fires with the most disastrous loss of life in the past 15 years in the United States were in mines equipped with mechanical ventilation, and with far more than ordinary efficiency in distribution of air to working faces and places. When timber is used underground, metal mines need mechanical ventilation for use in case of fire even where underground air temperatures are low, or where there is little or no harmful dust or there is essential freedom from gases.

The fire in the Magma mine at Superior, Ariz., on Nov. 24, 1927, had many interesting features.⁵ The mine has mechanical ventilation with electrically driven fans both on the surface and in the mine, and it is entirely probable that the fans and other features such as doors, additional shafts, crosscuts, etc., connected with the mechanical ventilation of the mine were responsible for the saving of the lives of at least 30, possibly as many as 40, of the 49 who were in the mine at the time the fire occurred. It is also probable that had the Magma shafts been supplied with fireproof or fire-resistant doors at each level leading from the shaft, at least some of those who were killed could have been saved or have saved themselves. The mine was fortunate in having two hoisting shafts and in having both main hoisting shafts downcast, which allowed the hoisting of most of the underground workers in pure air. The fire cost seven lives and probably several hundred thousands of dollars but it is entirely probable that the loss of life would have been around or over 40 and property loss in the millions had it not been for the aid given by the mechanical ventilation installation.

The Magma fire with its seven deaths recalls the Pennsylvania mine fire in Butte, Mont., in 1916, with 21 deaths; the North Butte fire in Butte in 1917, with 163 deaths; and the Argonaut fire in California in 1922, with 47 deaths. It should not be inappropriate at this time to call attention to some fire hazards in metal mines, with the hope that by pointing to such risks, they may be eliminated. Herewith are a few suggestions intended to aid in the avoidance of metal-mine fires or of disasters from such fires should they occur.

1. Every mine or working place in which more than 5 men work should have more than one opening for ingress and egress of both men and air. In mines reached by shafts more than 300 ft. deep there should be hoisting equipment in more than one shaft.

2. Every mine, coal or metal, employing 5 or more men should have mechanically controlled ventilation preferably through a fan located on the surface in fireproof housing and so arranged as to allow of prompt reversal of direction of air currents in case of necessity. The approach to the fan from the shaft should be fireproofed also.

3. No timbered shaft should act as both intake and return.

⁵ Described in *Engng. & Min. Jnl.* (1927) 124, 944.

4. Every opening leading from every shaft should have in it and reasonably close to the shaft a fireproof or fire-resistant door, so that by closing these doors air flow to or from the shaft may be excluded at any or all levels in case of necessity. Unless this system of doors is maintained (even if not used at ordinary times) at all times along all timbered shafts leading into deep mines there is danger of suffocation to all or practically all men in the mine in case of fire in the mine and especially in case of fire in or near the downcast shaft.

5. Ventilation doors should be solid, tight and fire-resistant; they should be equipped with a latch to keep them closed in case of reversing of direction of air and, where made automatic, the automatic feature should be positive as to closing the door rather than holding it open.

6. The shaft or shafts in which men are hoisted should be intake (or downcast).

7. Downcast shafts should preferably be lined with concrete or other absolutely fireproof material.

8. Underground electrical stations should be fireproof and all electrical equipment, wiring, etc., very carefully safeguarded as to fire.

9. No open lights, smoking, torches, or other flame ought to be allowed in any mine where timber or other inflammable material is used or found.

GASES AND OTHER HAZARDS

Up to 12 or 13 years ago, the candle was charged with the starting of most of the metal-mine fires in the United States and more recently electricity has had the blame; during the past year at least three serious metal-mine fires were caused by open flames. The Magma fire is now thought to have been started by a match or a cigarette; a recent technical article describes a fire in a shaft mine near Kimberly, Nev., where the ignition was by "burning torches" and where the crew in the shaft narrowly escaped suffocation; in the spring of 1927 an iron-ore mine in Pennsylvania had a fire which was started by ignition of the bark of a timber cap by a miner's open light. This fire cost considerable money, but, fortunately, no lives. These fires started by open flames indicate the danger of allowing open flame underground and it would appear that all types of open flame including open lamps, matches, torches, etc., should be excluded from all underground workings which have timber or other combustible material. During the year an open-flame miner's lamp ignited methane (the coal-miner's explosive gas) at the collar of a metal mine shaft in Michigan; the shaft was being dewatered and, as frequently occurs in such cases, explosive gas (probably from decomposition of timber under water) was encountered and two men were burned to death. There have been several instances of ignition of explosive gas (methane) with resultant explosions and deaths while dewatering metal-mine work-

ings in the United States in recent years; also there have been numerous instances of disastrous explosions of methane in driving tunnels in or near cities, notably at Kansas City in 1926 and also at Cleveland, Chicago, and other cities. There have also been several explosions in salt mines; generally open lights are the igniting agency whether in dewatering shafts, driving tunnels, or in working salt mines.

It would appear that all open flames should be prohibited in mines, and underground workers, whether in coal or in metal mines, should be prohibited from smoking and required to use only the safe, efficient electric storage-battery type of mine lamp, of which several types and makes are available. This method of lighting is little if any more expensive than the much more dangerous open-flame lighting systems and the light of the electric lamp is at least as great as can be expected from the open lights now in use.

A study of gas conditions in the Cripple Creek region, Colorado, recently completed by the United States Bureau of Mines, shows that the irrespirable gas long known to afflict several of the Cripple Creek mines is largely carbon dioxide and nitrogen with oxygen depleted, and it is proved that the gas is due to present-day, and day to day, chemical action of the mine waters on certain of the exposed mine strata. It is found that the maximum quantity of this irrespirable so-called strata gas entering any one mine is undoubtedly less than 3000 cu. ft. per min. as stored in the strata crevices and in certain porous rocks, that the gas can be held back in the strata by a mine atmosphere of less than 1 in. water gage, and that there are two daily periods when the mine air tends to flow into the strata. It is brought out that while the pressure ventilation method now used by certain mines with the mine atmosphere held under air pressure of $\frac{1}{2}$ to $\frac{3}{4}$ in. water gage will in general hold the gases into the strata, this system gives little opportunity to remove explosives fumes from working faces and there is no opportunity to utilize the cooling effect of moving air currents at hot working faces. It appears that use of fans of fairly large capacity with circulating currents would be much more effective than using small-capacity fans for holding a slight ($\frac{1}{2}$ to $\frac{3}{4}$ in.) water gage in the mine atmosphere. The records disclose that there are 35 well authenticated instances in the last 22 years of fatalities from this gas and it is probable that at least an equal number of men have fallen to death from ladders or into winzes or from raises or stopes, etc., because of the gas, and their death has been charged to the fall rather than to gas.

It seems possible that investigation may reveal that irrespirable gases in other metal-mining localities may also be due to present-day chemical action rather than to strata exhalations, as is the more general belief. Irrespirable gases or explosive gases have been found in metal mines in Cripple Creek, Colo.; Tintic, Utah; Tonopah, Nev.; Grass Valley, Calif.;

Park City, Utah; Picher, Okla.; Hibbing, Minn.; Gilpin, Colo.; Idaho Springs, Colo.; Butte, Mont.; San Juan district, Colo.; Birmingham, Ala.; Globe-Miami region of Arizona. Methane has been found in several salt mines in the United States and in the driving of a number of water and other tunnels for various cities in the United States such as Kansas City, Mo.; San Francisco, Calif.; Cleveland, Ohio; Chicago, Ill., and others. Irrespirable gases have been found in metal mines in England, Australia, France, New Zealand, the Philippines, and other foreign countries.

EXPLOSIONS IN METAL MINES

In addition to ignitions of explosive gas in metal mines in a number of regions of the United States, as above indicated, there were some dust ignitions in connection with working a high-copper iron sulfide ore in Arizona. Samples of this ore running over 40 per cent. sulfur were tested in a gallery by the United States Bureau of Mines at Pittsburgh and the dust ignited practically every time a dynamite shot was fired into it. Considerable volumes of sulfur dioxide and appreciable quantities of hydrogen sulfide were given off in these ignitions. The ignitions were definite but it was not proved whether or not the dust would propagate or continue an explosion in the manner that is so characteristic of our volatile coal dusts, and while definite pressures were recorded these pressures were by no means as great as in explosions of methane or of bituminous coal dust. It is probable that if blasting holes are tamped with clay or other incombustible these sulfide dust explosions will not occur, and if the face region is wet down by hose preparatory to blasting, the ignitions will probably be prevented.

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The Air-current Regulator

BY WALTER S. WEEKS,* BERKELEY, CALIF.

(New York Meeting, February, 1928)

IN coursing the ventilating air through a mine it is often necessary to restrict a comparatively open split in order that it may carry exactly the desired quantity of air. Such a restriction is known as a regulator, and usually takes the form of a partition with a hole in it. The quantity of air circulating is controlled by the size of the hole or orifice.

For example, it is desired to force a quantity q_1 cu. ft. per min. through split a (Fig. 1) and the quantity q_2 through split b . It requires a pressure

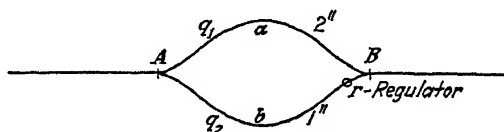


FIG. 1.—SPLITS WITH REGULATOR.

of 2 in. of water to force the desired quantity through a , and a pressure of 1 in. to force the desired amount through b . The pressure drop between points A and B must be 2 in. and a pressure of 1 in. must be destroyed in split b . This is done by inserting a regulator in b , which causes the velocity to increase in the orifice and results in a change from pressure-head to velocity-head. If the air, after passing the orifice, is slowed down in a turbulent manner, all of the velocity-head does not change back into pressure-head, so a permanent loss of static pressure results.

A classic formula for computing the area of an orifice to destroy a certain amount of static pressure has been handed down from early days:

$$a = \frac{0.0004Q}{\sqrt{h}}$$

This formula is based on the assumption that all velocity-head in the orifice is "destroyed," that is, does not reappear again as either velocity-pressure or static pressure. The derivation of the formula is as follows:

h = head lost in regulator in feet of air.

v = velocity in regulator in feet per second.

q = quantity circulating in cubic feet per second.

Q = quantity circulating in cubic feet per minute.

a = area of orifice in square feet.

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P = pressure lost in regulator in pounds per square foot.

i = pressure loss in inches water.

When a stream of air enters an orifice, it contracts so that the cross-sectional area of the stream is less than the area of the orifice. It is assumed that the area of the stream is 65 per cent. of the area of the orifice. It is also assumed that the density of the air is 0.075 lb. per cubic foot.

$$h = \frac{v^2}{2g}$$

$$q = 0.65av \quad v = \frac{q}{0.65a}$$

$$h = \frac{q^2}{0.42a^2 \times 64.32}$$

$$h = \frac{P}{0.075}$$

$$P = \frac{0.075q^2}{0.42a^2 \times 64.32}$$

$$P = 5.2i$$

$$i = \frac{0.075q^2}{0.42a^2 \times 64.32 \times 5.2}$$

$$i = \frac{0.075Q^2}{0.42a^2 \times 64.32 \times 5.2 \times 3600}$$

$$a = \frac{0.000385Q}{\sqrt{i}}$$

The derivation is based on three incorrect assumptions: (1) the density of air is constant; (2) the coefficient of contraction is constant; and (3) the entire velocity-head in the orifice is lost in shock.

It has been established in flow of fluids that when a moving stream enters a more slowly moving stream, the head lost in shock¹ is $\frac{(v_1 - v_2)^2}{2g}$

where v_1 is the faster velocity, v_2 the slower, and g the acceleration due to gravity. If the cross-sectional areas of the streams are known it is convenient to correlate the loss with the velocity in the duct. Let A_1 be the area of the faster stream and A_2 the area of the slower.

Then

$$A_1v_1 = A_2v_2$$

$$v_1 = \frac{A_2v_2}{A_1}$$

$$\text{Shock loss} = \frac{\left(\frac{A_2v_2}{A_1} - v_2\right)^2}{2g} = \left(\frac{A_2}{A_1} - 1\right)^2 \frac{v_2^2}{2g}$$

¹ Discussion of shock loss may be found in the following references:

W. H. Walker, W. K. Lewis and W. H. McAdams: Principles of Chemical Engineering, 1st ed., 90. New York, 1923. McGraw-Hill Book Co.

I. P. Church: Mechanics of Engineering, 721. New York, 1901. John Wiley & Sons.

For any given ratio of $\frac{A_2}{A_1}$ the loss of head due to shock is equal to a constant multiplied by the velocity-head in the slower stream.

The stream of air passing through the orifice contracts so that its area is less than the area of the orifice. The coefficient of contraction is not constant, as is assumed in the old approximate formula, but depends on the relative size of the drift and orifice.

If a is the area of the orifice and c is the coefficient of contraction, ca is the area of the faster stream. If A is the area of the drift, the shock loss is $\left(\frac{A}{ca} - 1\right)^2 \frac{v^2}{2g}$ where $\frac{v^2}{2g}$ is the velocity-head in the drift. If velocity-head is expressed in inches of water the expression gives the shock loss in the same unit.

If the ratio of drift to orifice be called R the shock loss in inches of water is

$$i = \left(\frac{R}{c} - 1\right)^2 \times \text{velocity-pressure in inches water.}$$

It may be possible that other losses of head occur in the regulator. Loss of head may take place in the action of contracting.

It was demonstrated in the Joplin and the Erie holder tests² that the drop in pressure through an orifice may be expressed as $K \frac{v^2}{2g}$ where $\frac{v^2}{2g}$ is the velocity-head in the duct and K is a factor that depends only on the ratio of the area of the duct to the area of the orifice.

The factor K contains the shock loss coefficient and the coefficients of whatever other losses may be present. Since K depends only on the ratio of duct to orifice and the shock loss coefficient depends only on the same ratio, $\left(\frac{R}{c} - 1\right)^2$ may be called K and the value of c calculated from experimental data. This results in what may be called a virtual value of c . If shock loss is the only loss, the virtual value is the true coefficient of contraction. If losses other than shock loss occur, the virtual value will be less than the true value. In any case, the virtual value of c used in the shock-loss formula for any given ratio will give the true drop in pressure in the orifice.

Virtual values of c were determined by Tom Berray and the writer for a number of different ratios. It was impossible to duplicate all conditions to be found in mines so it was considered that the use of the square duct and square orifices would be a sufficiently close approximation. The orifices used in these experiments were cut from cardboard 0.11 in. thick.

² For summary of tests see H. P. Wescott and J. C. Diehl: Measurements of Gas and Liquids by Orifice Meter. 2d ed. Erie, Pa., 1922. Metric Metal Works.

Air velocities were measured with the Pitot tube. The downstream static tube was placed downstream far enough to give maximum transformation of velocity pressure into static pressure. With each size of orifice two determinations for c were made with different quantities of air circulating. The results are given in Table 1.

TABLE 1.—*Virtual Coefficient of Contraction*

$\frac{\text{Area Duct}}{\text{Area Orifice}}$	10	5	4	3	2	1.75	1.5	1.25	1
Coefficient of contraction.	0.645	0.646	0.651	0.663	0.694	0.709	0.729	0.774	1
	0.644	0.647	0.648	0.660	0.694	0.710	0.725	0.774	1

The operation of the shock-loss formula for the purpose of determining the area of the opening in a regulator will now be illustrated.

$$i = \left(\frac{R}{c} - 1 \right)^2 \times \text{velocity-pressure}$$

$$\frac{R}{c} - 1 = \sqrt{\frac{i}{\text{velocity-pressure}}}$$

$$\frac{R}{c} = \sqrt{\frac{i}{\text{velocity-pressure}}} + 1$$

i is the water gage to be destroyed.

$$\text{Velocity-pressure} = \left(\frac{V}{4000} \right)^2 \times \frac{w}{0.075}$$

where V is the velocity in the drift in feet per minute and w is the actual weight of one cubic foot of air.

Determine the value of the right-hand member and then select corresponding values of R and c to satisfy the equation.

Example.—The area of a drift is 40 sq. ft.; quantity circulating 20,000 cu. ft. per min.; density, 0.075. It is desired to destroy a pressure of 1 in. water. What should be the area of the orifice?

$$V = \frac{20,000}{40} = 500 \text{ ft. per min.}$$

$$\text{Velocity-pressure} = \left(\frac{500}{4000} \right)^2 \times \frac{0.075}{0.075} = 0.0156 \text{ in. water.}$$

$$\frac{R}{c} = \sqrt{\frac{1}{0.0156}} + 1$$

$$\frac{R}{c} = 8.0 + 1 = 9.0$$

Selecting values of R and c to satisfy the equation and the table, we have

$$c = 0.645 \text{ and } R = 5.80.$$

$$\frac{R}{c} = \frac{5.80}{0.645} = 9.0$$

$$a = \frac{40}{5.80} = 6.90 \text{ sq. ft.}$$

Under certain conditions, the familiar approximate formula gives fairly accurate results, but under some conditions it breaks down entirely and may indicate that the area of the orifice to be used is larger than the drift itself. Its simplicity, however, is appealing, so an attempt has been made to develop a method by which the ratio of drift to orifice can be determined by means of the approximate formula and the true ratio read from a curve. The only assumption made is that the density remains constant at 0.075 lb. per cubic foot.

Using previous nomenclature, the head lost in the orifice according to approximate formula is $h = \frac{v^2}{2g}$. Correlate the loss with the velocity in the drift.

$$0.65av_1 = Av_2$$

$$v_1 = \frac{Av_2}{0.65a} = \frac{R_1v_2}{0.65} \text{ where } R_1 \text{ is the approximate ratio of drift to orifice.}$$

$$h = \left(\frac{R_1}{0.65} \right)^2 \frac{v_2^2}{2g}$$

$$\left(\frac{R_1}{0.65} \right)^2 = \frac{2gh}{v_2^2}$$

$$\frac{R_1}{0.65} = \sqrt{\frac{2gh}{v_2^2}} \quad [1]$$

The shock loss by the exact formula is

$$h = \left(\frac{R_2}{c} - 1 \right)^2 \frac{v_2^2}{2g} \text{ where } R_2 \text{ is the true ratio of drift to orifice.}$$

$$\left(\frac{R_2}{c} - 1 \right)^2 = \frac{2gh}{v_2^2}$$

$$\frac{R_2}{c} - 1 = \sqrt{\frac{2gh}{v_2^2}} \quad [2]$$

For any given conditions of area of drift, quantity flowing, and head to be destroyed, the right-hand members of the equations 1 and 2 are identical.

If in the exact equation 2 a value of R_2 is assumed and the proper value of c used, the value of the right-hand member is given for the speci-

fied conditions. For the same values of h and v_2 , then, the value of R_1 calculated by the approximate formula would be

$$\frac{R_1}{0.65} = \frac{R_2}{c} - 1$$

$$R_1 = 0.65 \left(\frac{R_2}{c} - 1 \right)$$

Assume different values of R_2 , insert the proper value of c and solve for R_1 . These corresponding values are plotted in Fig. 2. The abscissas are the ratios of the drift to orifice given by the approximate formula and the ordinates are the true ratios.

To determine the area of the orifice, solve the approximate formula for a . Divide the area of the drift by a . The result is R_1 . Enter the chart at the proper value of R_1 , pass vertically to the curve and the true ratio R_2 is read at the left. The area of the drift divided by R_2 is the true area of the orifice.

The example that was solved by the method of trial may now be solved by the use of the curve.

$$a = \frac{0.000385Q}{\sqrt{h}}$$

$$a = \frac{0.000385 \times 20,000}{1} = 7.7$$

$$R_1 = \frac{40}{7.7} = 5.2$$

From the chart R_2 is seen to be 5.80 and the true area of the orifice is $\frac{40}{5.80} = 6.90$.

It may be interesting to see how at times the approximate formula gives unintelligible results.

The area of a duct is 5.0 sq. ft., the quantity flowing is 10,000 cu. ft. per min., and it is desired to destroy 0.25 in. water.

$$a = \frac{0.000385 \times 10,000}{\sqrt{0.25}} = \frac{3.85}{0.5} = 7.60$$

The area of the orifice is then to be 2.6 sq. ft. larger than the drift.

$$R_1 = \frac{5}{7.60} = 0.66$$

From the curve it is seen that the true ratio is 1.48 and the true area of the orifice is $\frac{5}{1.48} = 3.38$ sq. ft. This may be checked by determining the loss by the exact formula.

$$V = \frac{10,000}{5} = 2000 \text{ ft. per min.}$$

$$\text{Velocity-pressure} = \left(\frac{2000}{4000} \right)^2 = 0.25 \text{ in. water.}$$

$$\text{Loss} = \left(\frac{R}{c} - 1 \right)^2 \frac{v^2}{2g}$$

$$\left(\frac{1.48}{0.73} - 1 \right)^2 \times 0.25 = 0.26$$

It was desired to destroy 0.25 in. water.

Notice that the curve (Fig. 2) passes through $R_1 = 0$, $R_2 = 1$. If no pressure at all is to be destroyed it is evident that no regulator must be inserted or, stating it another way, the ratio of drift to orifice must be one.

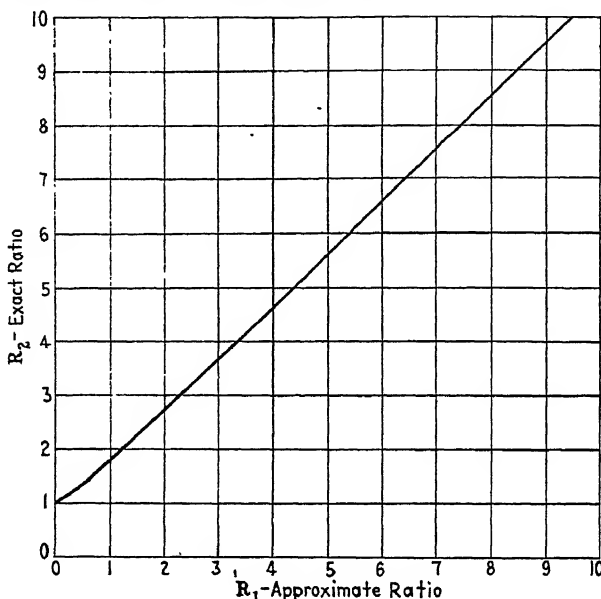


FIG. 2.—CONVERSION CURVE.

With this condition the approximate formula shows that the area of the orifice should be infinity.

$$a = \frac{0.000385Q}{\sqrt{0}} = \infty$$

Whence

$$R_1 = \frac{A}{\infty} = 0$$

The curve shows that when $R_1 = 0$, the correct ratio, R_2 , is one.

DISCUSSION

G. E. McELROY, Pittsburgh, Pa. (written discussion).—Mine ventilation is concerned essentially with the pressure losses caused by but one type of orifice and that is the square-edged, thick-plate orifice. The air-current regulator is an orifice of this type and represents one of the two primary conditions of installation as affecting pressure losses—the orifice inserted in the air duct; in this case, the mine airway. The other primary condition of installation is that of the orifice placed on the discharge end of an air duct,

as typified by the use of orifice plates in fan-testing work, where the ratio of orifice area to fan discharge area is termed "ratio of opening" and used as a basis for plotting generalized fan performance data. For mine ventilation purposes it is desirable to be able to determine approximate pressure losses for this type of orifice for both of these primary conditions of installation. It happens that rather meager data secured by Mr. Richardson and myself on the one condition³ so closely confirms Professor Weeks' more complete data on the other primary condition that it seems permissible to correlate the two sets of data in order to develop a consistent, or faired-up, set of values for both conditions.

Analysis of the pressure loss results that we secured in Butte showed that the static pressure at four $\frac{7}{32}$ -in. square-edged circular orifices installed on the end of a pipe 18 in. in diameter bore a quite constant ratio to the difference in velocity pressure as between the orifice area and the pipe area. On receipt of Professor Weeks' data, I computed similar ratios for his orifices as used on the end of a duct, for which condition the shock loss is $\left(\frac{R}{C} - 0\right)^2$, or $\left(\frac{R}{C}\right)^2$, times the pipe velocity pressure instead of $\left(\frac{R}{C} - 1\right)^2$ as developed by him for orifices inserted in the line. The static pressure at the orifice is one pipe velocity pressure less than the total pressure and is therefore $\left(\left(\frac{R}{C}\right)^2 - 1\right) \times$ the pipe velocity pressure. These computations show a similar, and practically equal, quite constant ratio of static pressure to difference in velocity pressures. Values of this ratio, as derived from both sets of test data, are given in

TABLE 2.—*Coefficients of Contraction and Related Data for Square-edged, Thick-plate Orifices*

$\frac{1}{R}$	R	Author's data		Weeks' data		C based on $\frac{S. P.}{D. V. P.} =$	
		$\frac{S. P.}{D. V. P.}$	C	$\frac{S. P.}{D. V. P.}$	C	2.45	2.5
0.100	10.00			2.42	0.644	0.641	0.634
0.121	8.27	2.32	0.659			0.644	
0.200	5.00			2.45	0.647	0.647	0.640
0.241	4.15	2.47	0.648			0.650	
0.250	4.00			2.46	0.650	0.651	
0.300	3.33					0.657	0.650
0.333	3.00			2.45	0.661	0.661	
0.400	2.50					0.672	0.665
0.487	2.05	2.39	0.699			0.689	
0.500	2.00			2.44	0.694	0.692	0.686
0.572	1.75			2.46	0.709	0.709	
0.600	1.67					0.720	0.714
0.667	1.50			2.61	0.727	0.744	
0.700	1.43					0.758	0.753
0.732	1.37	2.47	0.770			0.773	
0.800	1.25			2.86	0.774	0.811	0.806
0.900	1.11					0.885	0.882

³ G. E. McElroy and A. S. Richardson: Experiments on Mine Fan Performance. U. S. Bur. Mines *Technical Paper* prepared for publication.

Table 2 under the designation $\frac{S. P.}{D. V. P.}$ for the corresponding values of R and C as used in this paper. By inspection of the table, it can be seen that the values of this constant are in close agreement, and, for the intermediate range where the probable accuracy of test work is greatest, are practically constant at 2.45. Analysis of data on other orifices have shown similar relations; that is, a practically constant ratio over the range where good test results would be expected and small variations from this value for the extreme orifice ratios. For instance, in the same set of Butte

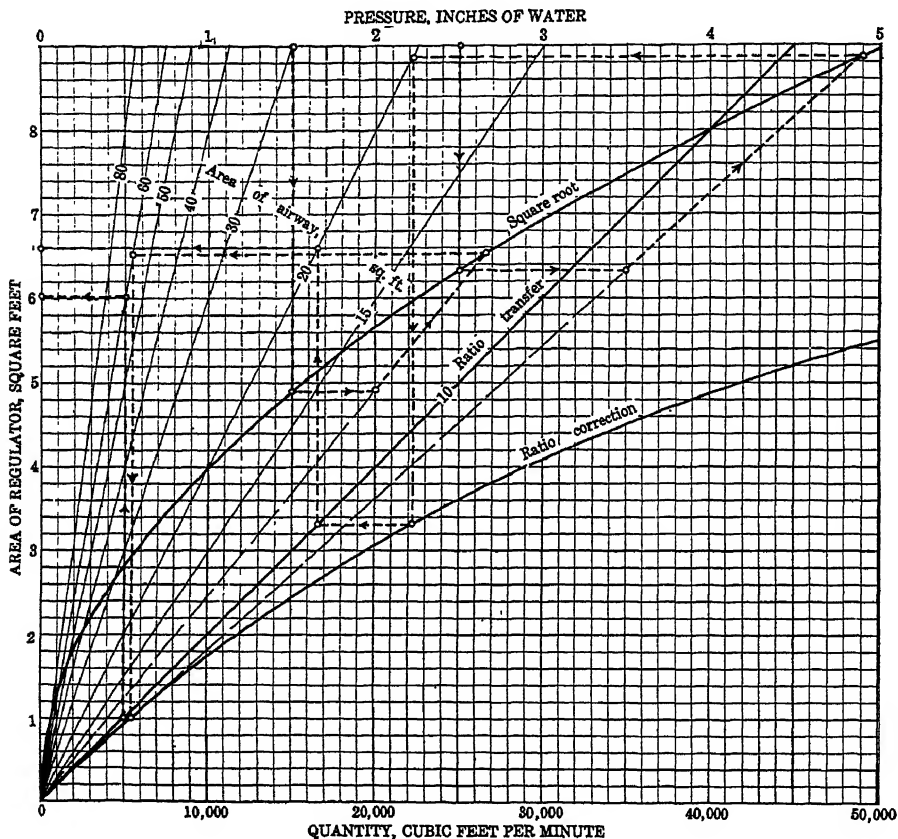


FIG. 3.—GRAPH FOR DETERMINING AREA OF ORIFICE REGULATOR IN MINE AIRWAY REQUIRED TO CAUSE A DEFINITE PRESSURE LOSS FOR A DEFINITE VOLUME OF FLOW.

experiments, we got a value of 0.52 for a 20° cone orifice in an 18-in. dia. pipe, and two, sets of standard data for thin-plate orifices give constants over a wide range of orifice conditions that vary but little from 2.70.

In consideration of the fact that mine-orifice conditions are not exactly standard the use of the constant 2.45 seems an unnecessary refinement, and I propose to use 2.5 as a basic, easily remembered figure for the pressure loss at square-edged, thick-plate orifices. With this basic figure for the ratio of the static pressure at an orifice on the end of the line to the difference in velocity pressures as between orifice area and duct area, it is possible to compute a shock-loss coefficient of contraction for either of the two primary conditions of installation. Coefficients computed on this 2.5 basis

are compared in the table with those resulting from the two sets of tests and those of the more exact 2.45 basis. The differences between the various sets are so slight as to have no practical significance as applied to air-current regulator problems or to problems in fan performance involving ratios of opening or velocity pressure ratios.

Professor Weeks has developed a mathematical method of determining the proper ratio of regulator area to airway area and a simpler method of doing the same thing that is partly mathematical and partly graphical. In connection with the latter it should be noted that the graph shown in Fig. 2 is based on the use of a coefficient

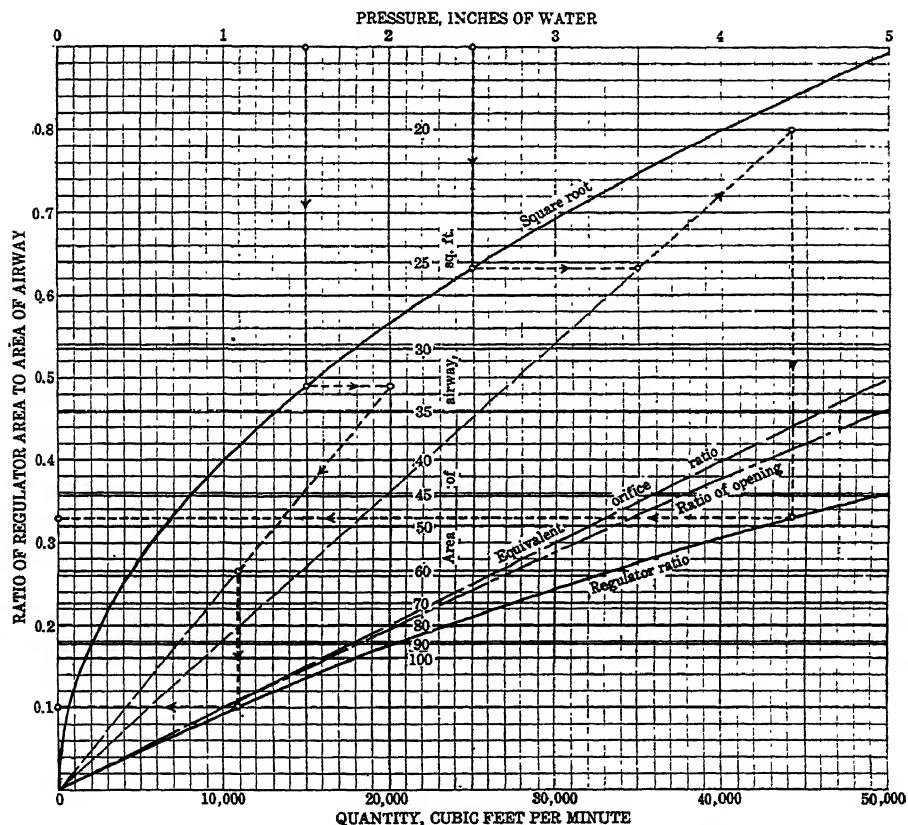


FIG. 4.—GRAPH FOR DETERMINING RATIO OF AREA OF ORIFICE REGULATOR IN MINE AIRWAY TO AREA OF AIRWAY REQUIRED TO CAUSE A DEFINITE PRESSURE LOSS FOR A DEFINITE VOLUME OF FLOW.

of contraction of 0.650 in the equivalent orifice formula and that the corresponding constant in the normal form of this formula is then 0.000385 instead of 0.0004 as commonly used. The coefficient of contraction corresponding to a constant of 0.0004 is 0.624 and these values are used in the two charts shown in Figs. 3 and 4, both of which are based on shock-loss coefficients of contraction derived on the 2.5 basis.

In Fig. 3, the area of the regulator can be determined directly for known pressure loss and quantity data but the steps are rather involved; some of the intersections occur at very acute angles and there is a possibility of confusing the separate sets of lines used. For these reasons the form shown as Fig. 4 was developed as somewhat

simpler, although it determines directly only the ratio of regulator area to airway area and one simple mathematical step is involved in determining the actual area of the regulator from its ratio to the airway area. The steps involved in solving problems by the aid of each chart and the steps involved in their construction will be explained.

To use Fig. 3, enter the chart at the top edge at the known pressure; pass vertically downward to intersection with the "square root" curve; pass horizontally to intersection with a vertical line through the known quantity as shown on the lower edge; pass a straight line through this point and the zero point of the chart (lower left corner); follow this slanting line to intersection with the "square root" curve; pass horizontally to intersection with the estimated position of a slanting line corresponding to the known area of airway; pass vertically to intersection with "ratio correction" curve; pass horizontally to "ratio transfer" curve; pass vertically to intersection with estimated position of area of airway line; pass horizontally to scale on left and read area of regulator.

In practice, the sequence of steps is not as difficult to follow as this detailed explanation would indicate, as can be demonstrated by following the steps through for actual problems. Two of these are indicated by dotted lines on the graph. In one, the regulator is to cause a pressure loss of 1.5 in. with a flow of 20,000 cu. ft. per min. through an airway of 60 sq. ft. and the size of the regulator required is found to be about 6.0 sq. ft. In the other example traced on the chart, the regulator is to cause a pressure loss of 2.5 in. with a flow of 35,000 cu. ft. per min. through an airway of 20 sq. ft., and the size of the regulator required is found to be 6.6 square feet.

The steps involved in the construction of this chart are as follows: Pressures (i) to scale along the top edge are plotted against their square roots (\sqrt{i}) to a scale, not shown, along the left edge. Quantities (Q) are scaled along the lower edge. Then the coordinates \sqrt{i} and Q determine one point on straight lines of equal resistance for which another set of coordinates are 0, 0. Then the scale of areas in square feet was placed on the left and values of equivalent orifice areas were plotted against various equal resistance values, which, through a proper selection of scales, resulted in a curve coincident with the square-root curve. A scale of ratios of equivalent orifice area to area of airway, not shown, was established along the lower edge, and area of airway lines drawn so as to intersect equivalent orifice areas at the proper points. An exactly similar scale, not shown, was established along the left edge to denote true ratio of regulator area to area of airway. Values of equivalent orifice area ratios for certain values of true orifice area ratios were computed and plotted against each other, resulting in the "ratio correction" curve shown which is similar to that shown in Fig. 2 of Professor Weeks' paper, except that ratio of regulator to airway is plotted instead of ratio of airway to regulator. In order to get from true ratio to actual areas, it was necessary to use these true ratios on the lower scale, which is accomplished by the 45° line marked "ratio transfer."

To use Fig. 4, enter the chart at the top edge at the known pressure; pass vertically downward to intersection with the "square root" curve; pass horizontally to intersection with a vertical line through the known quantity scaled along the lower edge; pass a straight line through this point and the zero point of the chart (lower left corner); follow this slanting line to intersection with the estimated position of a horizontal line corresponding to the known area of airway; pass vertically to intersection with the "regulator ratio" curve; pass horizontally to scale on left and read ratio of regulator area to airway area. Multiplication of the latter by the former gives the actual area of the regulator required.

Solution of the same two problems traced on Fig. 3 are also indicated by dotted lines on Fig. 4. For the one, a pressure loss of 1.5 in. for a flow of 20,000 cu. ft. per min. in an airway of 60 sq. ft. requires a ratio of regulator area to airway area of 0.10; that is, an area of 6.0 sq. ft. For the other, a pressure loss of 2.5 in. for a flow of

35,000 cu. ft. per min. in an airway of 20 sq. ft. requires a ratio of 0.33; that is, an area of 6.6 square feet.

The steps involved in the construction of this chart differ in some particulars from those for Fig. 3, as follows: Pressures, quantities and square root of pressures were similarly scaled along the top, bottom and side respectively. The square-root curve and lines of equal resistance were obtained in the same manner as in Fig. 3. A horizontal line to represent 20 sq. ft. was arbitrarily placed coincident with $\sqrt{i} = 2$ and the scale of ratios was established on the left. Then the position of the "regulator ratio" curve was plotted against quantities obtained by solving equations of the form $i = \frac{\text{Constant}}{A^2} Q^2$ for which constants for the even ratios were computed from the orifice data. A solution was then made for the \sqrt{i} for other values of area of airway and horizontal lines were drawn through these values. Solutions were also made for quantities against ratios of area for the equivalent orifice formula $\sqrt{i} = \frac{0.0004}{a} Q$ and for ratio of opening, so that if the intersection of the last vertical line in the method of solution for regulator ratios is taken at the "equivalent orifice ratio" or "ratio of opening" curve instead of the "regulator ratio" curve, and followed horizontally to the scale on the left, the result will be the ratio of equivalent orifice area to area of airway for the one and the actual ratio of area of orifice on the end of a duct to the area of the duct for the other. Inspection of these three curves indicates that while the differences are small for small orifice ratios, they increase rapidly as the orifice ratio increases.

Inspection of these charts and an estimate of the normal range of values of pressures, quantities and areas involved in air-current regulator problems, indicates that in most cases the ratio of regulator area will fall at or below the value 0.2; that is, in the range for which the equivalent orifice method of computation is reasonably accurate. The large spread between the "regulator ratio" curve and the "ratio of opening" curve in Fig. 4 results from the large difference in pressure losses for the same orifice depending on whether it is inserted in the air duct or on the end of the duct. The small spread between the "ratio of opening" curve and the "equivalent orifice ratio" curve is due to differences in pressure losses between those resulting from a variable coefficient of contraction and those calculated from a constant coefficient. It is thus seen that, in applying the equivalent orifice formula to air-current regulator problems, a much greater error results from considering the orifice to be at the end of a duct instead of actually in the duct than results from the assumption of a constant coefficient of contraction.

The question of density, mentioned by Professor Weeks as another failing of the equivalent orifice formula, is of relatively little importance in regulator computations, since such computations, concerned as they are with the balancing of the resistances of parallel circuits, may be carried out on the assumption of standard density. The two charts I have presented are based on a standard density of 0.075 lb. per cu. ft. If the pressure and quantity at some other density are known, it is necessary to correct the pressure only by multiplying it by the ratio of 0.075 to the known density, since for constant quantity, pressures vary directly as the air density.

Attention should be called to the fact that, under ordinary conditions of application, there is no need of any high degree of accuracy in air-current regulator problems since mine resistances are continually varying and the actual area at any point in a mine airway is often quite different from the nominal area on which the calculations are most often based.

G. S. RICE, Washington, D. C.—There has been much development going on in ventilation since our meeting of a year ago, both for metal mines, coal mines and

tunnels and increasing knowledge of the physics of ventilation. Research in ventilation has been carried on by various agencies, especially by the mining departments of the universities of Illinois and California, by the Bureau of Mines and cooperating mining companies. Somewhat parallel work has been going on in Great Britain and in Germany.

Prof. Henry Briggs of Edinburgh, in a series of lectures which he recently gave at London University, made a comparison between the weight of air moved in ventilating coal mines with the weight of the coal hoisted from the respective mines. It was rather surprising to learn that the weight of air in circulation in the course of 24 hr. was from 5 to 10 times greater than the weight of the coal produced. As concerns the proportion of power used at coal mines of the total power consumed, Professor Briggs quoted from a University of Illinois bulletin on an investigation made, under direction of Professor Callen, of the ventilation of 40 mines in which 22.45 per cent. of the total power used at the mines represented the power consumed by the fans. Professor Briggs says about the same proportion prevails in Great Britain, but that the costs are more in Great Britain because of the deeper mines and other conditions.

Studies were made in three groups of mines of different weekly production. In the first group with an average weekly production of 14,000 long tons (2240 lb.) the average horsepower in the air (he does not give the consumption of horsepower of the motors), in the 10 mines of the group was 220 hp., and the highest was 394 hp. The weight of air circulated in 24 hr. was five times that of the coal hoisted. In a group of nine mines which produced from 7000 to 14,000 tons, the average horsepower in the air was 136. The ratio of weight of air to the coal hoisted was 5.9 per cent. to 1. In a group of nine smaller mines, which produced under 7000 tons per week, the average horsepower in the air was 49, and the ratio of weight of air to weight of coal was 6.2 to 1.

Professor Briggs attempts to estimate for all the coal mines of Great Britain, and as you know, the annual production of that country is less than half that of this country. He figures that there are 80,000,000 tons of air circulated in 24 hr., and the annual cost of the mine ventilation amounts to \$10,000,000. He estimates that the average cost of ventilation is 4 c. per ton of coal produced and that at some of the collieries the cost for power of ventilating is about \$50,000 per annum.

He quotes from Mr. Davis of South Wales. In 11 collieries the cost varied from 3.2 c. to 18 c. per ton of coal hoisted. To us these figures seem to be high.

More of such investigations are needed to determine the costs and locate unnecessary losses. Evidently the losses are large. The average mining man does not appreciate the amount of money lost in a leak, yet in many mines only a quarter of the air that leaves the fan reaches the faces. When the mine owners begin to appreciate that air leakage means continued loss of money they will appreciate the advantage of intensive ventilation investigations made in their mines to stop the leakage of air and dollars.

There is one problem which we have in metal mining, but fortunately not in coal mining, in this country, and that relates to the high rock and air temperatures. I noticed in an interesting article recently published in Great Britain that the conclusion seemed to be that after all the high-temperature problem was one of increased ventilation, to carry away the heat at the working places.

A. C. CALLEN, Urbana, Ill.—At the time that I suggested that I would submit a discussion on this paper, Professor Weeks asked me if I could run a series of experiments prior to the meeting to either substantiate or refute the values of coefficients which he had determined. Owing to the amount of time required, I did not get the results early enough, or in sufficient detail, to analyze them as I would like to have done. I worked over some data on the flow of water through orifices and found that

in general the coefficient of contraction as determined by Professor Weeks' method was considerably less in the majority of cases than the values he gave here for air. Those values, however, were for circular orifices in circular pipes. Professor Weeks' values are for square orifices in square pipes, so that there might perhaps be a difference expected from that source.

My plan was to rig up a duct and use orifices of a square, circular, and rectangular shape, in a square pipe, to see if I could confirm Professor Weeks' results and see how well the circular orifices in the square duct checked with water. D. R. Mitchell and I ran about 100 tests, and I will briefly give the results.

In general, we confirm Professor Weeks' coefficient of contraction almost exactly. We ran an orifice with a ratio of 1.33, and plotted a curve with that as the low point and found that our points all checked very well with Professor Weeks'.

For two circular orifices in a square pipe the coefficients were 2 or 3 per cent. below, roughly, the values for a square orifice in a square pipe. The rectangular orifices checked fairly well with the results for square orifices, but we did not have sufficient time to run a large enough number of tests really to make a definite conclusion on this matter. These tests were made in a square duct, 10.4 in. on the side, of galvanized iron. All the joints were either soldered or very carefully calked up with a modeling clay, so there were absolutely no leaks in the system. The velocity measurements were made by means of the Pitot tube. The orifice was placed approximately 15 pipe diameters from the Pitot tube station, and the drop in pressure was taken over about 25 ft. of duct, which included the orifice.

Previous tests were run on the drop in that 25 ft. of duct without orifices, so as to get the drop caused by various velocities. From these drops a curve was plotted, enabling corrections for the different quantities to be subtracted from the total drop when using an orifice, the remaining cost head being attributed to the orifice.

The symmetry of pipe and orifice will probably play a considerable part in determining the coefficient of contraction. If we have a circular pipe and circular orifice, I think we may perhaps get about the same coefficient of contraction as if we had a square pipe and a square orifice; but if a square orifice is put in a circular or a rectangular pipe, I am not quite sure that would follow the curve which might be plotted from Professor Weeks' results. I would say, however, that I am pretty well of the opinion that Professor Weeks' results could be used in either square or rectangular ducts, with either square or rectangular orifices, and the error would probably not be over 2 or 3 per cent.

G. S. RICE.—In glancing over Mr. McElroy's discussion of Professor Weeks' paper, I noticed that he has carried out some tests at Butte. He gives comparable data. Can you analyze that offhand? The results seem to check up reasonably.

A. C. CALLEN.—It would appear that Mr. McElroy ran four tests with a circular orifice and circular pipe, and the coefficients which he obtained seemed to check very closely, although the ratios were not quite the same as Professor Weeks' values.

G. S. RICE.—There seems to be a close correspondence in the upper range.

A. C. CALLEN.—They are very close. In Professor Weeks' values, for 1.25 ratio he gives 0.774 and Mr. McElroy found 0.770. At another place Mr. McElroy got 0.648. It looks as though they corresponded very closely indeed, if that c is the same c . If that is so, it looks as though the symmetry of the circular duct and orifice might have something to do with the results on the square pipes being almost the same.

H. P. GREENWALD, Pittsburgh, Pa.—Mr. McElroy obtained his data while doing certain test work in the metal mines at Butte. The data are along the same lines as Professor Weeks' and agree very well with his. The agreement is quite satisfactory

and much better than would be expected on large-scale experiments. Mr. McElroy has developed in this discussion a graphic method of solution of the problem for which Professor Weeks has a mathematical method, and the development of that graphic method, and the giving of the example and the graphs, constitute three-quarters of the discussion, but so far as discussion of Professor Weeks' paper goes, he has really confirmed it with his own data.

C. EVANS, JR., Scranton, Pa.—In your experiments, did you place your orifices symmetrically in the duct? Would that have any effect upon the experiment?

A. C. CALLEN.—I cannot answer the latter question exactly. The center of the duct and the center of the orifices coincided, and Mr. McElroy's did also. In work done with water and circular orifices at the University of Illinois some few years ago, tests were made with some eccentricities of orifices, and my recollection is that a moderate shift of the center of the orifice did not seem to affect the results very much.

A. W. HESSE, Nemaquin, Pa.—What effect does the difference in the temperature of the air have on the formula that Professor Weeks has developed, especially in regard to his coefficient of contraction? I have noticed that during the winter months we get considerably more air than we do during the summer months from the same number of revolutions per minute.

A. C. CALLEN.—The temperature of the air would have no effect because the density cancels out in obtaining the coefficients. The other point is that more air is obtained in the winter than in the summer. Natural ventilation is probably the influencing cause there.

W. S. WEEKS (written discussion).—The coefficient of contraction given in this paper is, in reality, not the true coefficient of contraction that would be obtained by photographing the stream of air. It is, as I have called it, a virtual coefficient which, if used in the shock loss formula, will give the drop in pressure between the two static taps leading to the manometer.

I see no way of separating out the turbulence due to what we call friction in the pipe section from that due to the shock of the faster stream impinging on the slower. When the orifice is inserted, the stream lines and velocities are entirely different from those encountered in normal flow through the pipe so the effect of the pipe on the drop seems to be indeterminable. In my coefficients the loss in the pipe between the two places where pressure is measured is reflected in the coefficient.

Attention seems to be focussed on the exact values of the coefficients which will vary with variations in the construction of the orifice. I think that if it is agreed that the method of computing the size of an orifice to "do away" with a certain amount of static pressure is correct a standard orifice for mines should be designed, the coefficients should be determined and a conversion curve plotted similar to Fig. 2; or a better solution should be devised.

The particular method for determining the correct ratio was used because the equivalent orifice formula is an old friend and it is easier for some to compute the ratio with the old formula and read the true ratio from a curve than to learn an entirely new method.

Propeller Fan Computations

BY F. ERNEST BRACKETT, CUMBERLAND, MD.

(New York Meeting, February, 1925.)

THE simplicity of the propeller or disk fan, its small size and low cost, has, in recent years, led to an extended use of ventilators of this type at mines where only slight pressure is required. On this account the author of this paper has prepared for the new edition of Peele's Mining Engineers' Handbook a more accurate method of computing the required size, power, and so forth, than was possible for the first edition. The main object of this paper is to set forth the data collected in the course of that work, and to discuss the subject with more latitude than would have been advisable in a handbook, even if space had been available.

THEORY OF FANS

A great deal has been written about the theory of fans from Rankine's time down to the present. Nevertheless a rigid mathematical analysis of the propeller fan seems to be wanting. A general idea of the mechanics and difficulties in the way of analysis may be gathered from the following:

The useful pressure and velocity effects of the oblique impact of the vanes on the axial current would evidently be nil if the blades were perpendicular to the plane of rotation, or if they were of such helicoidal form that the screw pitch multiplied by the speed represented an axial velocity agreeing with the actual velocity. Somewhere between these zero angles there is a pitch which, under given conditions of flow, would produce the maximum useful effect. A true helicoidal blade can be so arranged that it would offer no retarding effect on the air current at any part of the wheel, provided the axial velocity be assumed the same in all parts of the cross-section of the air course, but if plane surfaces are used, it is possible, under some conditions, that the air would be retarded at the middle parts of the wheel while it was accelerated at the circumference. Whatever be the form of vane, it seems evident that the tips are far more effective than the center of the wheel, because of their higher velocity. For this reason some investigators believe that a backward current forms through the middle of the wheel and that part of the air is circulated in a radial axial loop. Ignoring the possible existence of this eddy and the friction of the walls of the air course; and assuming the axial velocity of the air through all parts of the wheel, some rational analysis of the impact might be made according to the theory of deflecting surfaces, were it not

for the unstable velocities developed on the lee side of the fan. The most important of these are the residual rotational velocities of the air which must immediately tend towards the formation of a free vortex, causing a partial vacuum at the center and increasing the counter pressure at the circumference. Thus the primary axial urge of the wheel is modified, and doubtless improved, by the centrifugal effects back of it; which accelerate the central axial velocities where the wheel is inactive and retard the flow at the circumference. When air is prevented from passing the fan, by placing a diaphragm in the duct back of the wheel, this centrifugal action causes the air to circulate *forward* through the center and backward in the clearance space outside the tips at the circumference.

In practice, the angle of the tips of the vanes with the plane of revolution is usually from 30° to 40° and the mean axial velocity much less than the tangential speed of the tips, and less than the axial urge which the screw pitch would develop. Also, the combined pressure effect of both impulse and centrifugal force is less than the head corresponding to the peripheral velocity.

The Quasi Theory

The quasi theory of propeller fans applies to the usual type, operated within ordinary speed limits. It is based partly on theoretical and partly on empirical grounds. According to this theory, the total head (H) developed, is considered proportional to the head corresponding to an axial velocity found by multiplying the screw pitch at the tips of the vanes by the number of revolutions. As this pitch is, in practice, confined within fairly narrow limits, H may be considered approximately proportional to the square of the tangential velocity u^2 . The part of H absorbed by the real axial velocity, v , is $v^2 \div 2g$. The remainder appears as pressure caused by resistance to the flow. Suppose this resistance to be represented by its equivalent orifice, and let r be the ratio of the area of that orifice to the area of the fan, A . The velocity through the orifice then is $vA \div rA = v \div r$; and the pressure required to overcome the resistance is, approximately, $v^2 \div 2gr^2$; that is, for equal values of r , this part of H is proportional to the square of the axial velocity. Hence, since both parts of H are proportional to the square of the real axial velocity, H varies as v^2 ; and, since it also varies as u^2 , v varies as u under equal values of r .

The brake horsepower for propeller fans can not be considered as closely related to the useful work, since the eddies produced when little or no air is passing exert a powerful counter-torque against rotation. This power is more nearly proportional to the speed of the tips of the vanes, total pressure, and wheel area than to useful work.

Stated concisely, the results of the quasi theory are: With the same ratio between the equivalent orifice and the area of the fan (1) the pres-

sure developed bears a constant ratio to the head corresponding to peripheral velocity, (2) the axial velocity bears a constant ratio to the peripheral velocity, and (3) the brake horsepower bears a constant ratio to the continued product of the area, total pressure, and tangential speed. These ratios, for various kinds and sizes of fans under different operating conditions, are given in Table 1 in columns 1, 7, 8, and 11.

FAN CHARACTERISTICS

Tests are conducted by attaching the fan to a tube of the same diameter; in which a diaphragm, carrying an adjustable resistance orifice, has been placed. Measurements consist in ascertaining the pressure of the air inside the test tube between the orifice and the fan, the volume of air passing, and the power consumed; corresponding to various speeds and orifice areas. The results, for any one speed of rota-

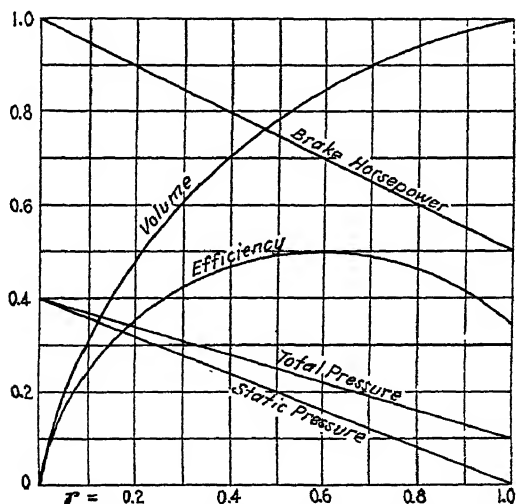


FIG. 1.—COMPARISON OF SPEEDS AND SIZES, USING RATIOS FOR COORDINATES.

tion, may be plotted by using the test results as ordinates and the corresponding areas of the orifice for abscissas. A more useful and common form of graph is formed by using ratios instead of absolute values for the coordinates thus permitting a more ready comparison between results for different speeds and sizes. Fig. 1 is a sketch of a graph made in this way. The horizontals are $r =$ orifice area \div fan area. The verticals for the pressure curves represent pressure divided by the "peripheral velocity pressure" or $h \div \frac{u^2}{2g}$. The curves of volume and horsepower are percentages of the maximum values of those quantities at $r = 1$ and $r = 0$ respectively.

The volume curve is a direct function of r , being convex upward and passing through the origin. Brake horsepower is an inverse function, being greatest when $r = 0$ and decreasing as the volume increases. Hence a fall or stoppage in the air course does not tend to create excessive speed at the fan. The efficiency graph is convex upward, and shows a maximum at approximately $r = 0.60$. There are two pressure graphs; one representing the static pressure and the other the total pressure which includes the velocity head. Both begin at the same point on the ordinate $r = 0$, since at that point there is no velocity, and fall with increasing divergence until, at $r = 1$, the static graph reaches zero and the total pressure registers a head corresponding to axial velocity when the fan is running "open" or without resistance.

INTERPRETATION OF THE PRESSURE GRAPHS

It is essential to distinguish carefully the significance of these graphs in relation to the observed water gage and the hypothetical mine resistance or useful work. With a blower, the static graph represents the observed water gage, and the total-pressure graph equals the sum of the frictional resistance of the mine and the head necessary to develop entrance velocity at the intake. These graphs are identical whether the fan is installed as a blower or an exhauster, and the total pressure equals

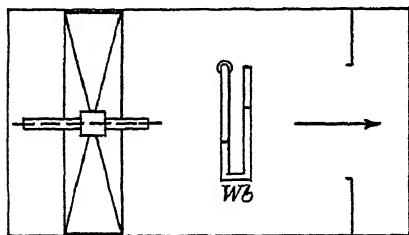


FIG. 2.—DIFFERENCE IN STATIC PRESSURE BETWEEN COMPRESSED AIR IN ORIFICE CHAMBER AND OUTSIDE ATMOSPHERE, MEASURED BY WATER GAGE W_b .

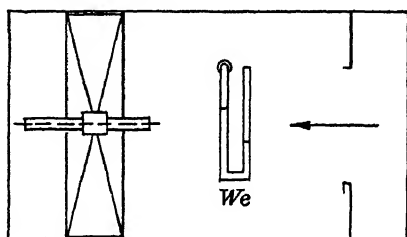


FIG. 3.—DIFFERENCE IN STATIC PRESSURE BETWEEN AIR AT AXIAL VELOCITY ON BOTH SIDES OF FAN, MEASURED BY WATER GAGE W_e .

mine friction and velocity head in either case; but if used as an exhaust fan the total-pressure graph represents the water gage. This is made clear by a study of Figs. 2 and 3.

Let H be the difference in static pressure, expressed in feet of air head, between air in the fan duct, at axial velocity v , before and after it passes the fan. In an experiment made with a blower, as in Fig. 2, the water gage W_b measures the difference in static pressure between compressed air in the orifice chamber and the outside atmosphere. This is less than H by a quantity equivalent to the head necessary to develop the axial velocity v . Hence, with a blower, $H = W_b + \frac{v^2}{2g}$ = total pressure ordi-

nate = sum of mine friction and velocity heads. In an exhausting experiment, Fig. 3, the water gage W_e measures the difference in static pressure between air at axial velocity on both sides of the fan, because the exit velocity is absorbed by fluid friction and does not produce any difference in pressure between the tube and the outside air. Hence $W_e = H$. The quantity $h_o = W_b = \left(W_e - \frac{v^2}{2g} \right) = \left(H - \frac{v^2}{2g} \right)$ is called the zero-zero pressure of the ventilator, and expresses the pressure which would be produced between two large bodies of quiescent air. It has been used as a basis in comparing ventilating machines to obviate confusion as to the method of reading the gage. Nevertheless, the velocities produced represent potential energy, a part of which can usually be converted into useful work by properly arranging the plant. Hence "plant efficiency" will sometimes differ from the "fan efficiency" computed by using h_o as the useful pressure.

DATA ON PROPELLER FANS

Table 1 has been compiled principally from the manufacturers' trade circulars. To this have been added a few data from graphs and from Kent's Pocketbook. In selecting the data an effort has been made to get a general "cross-section" of results from fans differing widely in size and operating conditions. In this table the figures given in columns 6, 7, 8, 10 and 11, are almost entirely the results of computations by the author on the given data. In cases where the value of r was not stated, its value was computed by using a *vena contracta* coefficient of about 0.9. This figure, arrived at by computations on data where the ratio was given, is probably too large for the lower orifice ratios. In comparing the efficiencies it should be borne in mind that these include mechanical friction, and that some of the fans use roller bearings. Hence high efficiency does not necessarily indicate superior design from a hydrodynamical standpoint.

EMPIRICAL FORMULAS

The following formulas are an elaboration of the quasi theory, based on fan graphs and the data given in Table 1.

- Let: D = diameter of wheel in feet,
 N = number of revolutions per minute,
 Q and q = air delivered, cubic feet per min. and per second,
 A = area of fan = $0.7854 D^2$,
 v = mean axial velocity, feet per second = q/A ,
 W = water gage corresponding to H , inches,
 H = total pressure, feet of air, = $5.2W/d = 69.3W$,

TABLE 1.—Data on Propeller Fans

1 Orifice Ratio r	2 Diam., Ft.	3 Speed, R.p.m.	4 Volume, Cu. Ft. per M., Q	5 Water Gage, In.	6 W	7 $\frac{2gh}{u^2}$	8 $\frac{v}{u}$	9 B.hp.	10 Effic. E	11 $\frac{B.hp. \times 10^4}{60uAW}$
0.	4.0	598	0	1.10	1.10	0.31	0.	4.24	0.	0.41
0.	8.0		0			0.43	0.		0.	
0.25	4.0	598	10,200		0.96	0.27	0.106	3.52	0.365	0.39
0.25	8.0	305	65,000		1.00	0.28	0.170	26.00	0.54	0.68
0.27	1.5	1,150	1,530	0.75	0.80	0.44	0.160	0.73	0.26	0.95
0.27	4.0	430	10,800	0.75	0.80	0.44	0.160	5.15	0.26	0.95
0.27	8.0	215	43,500	0.75	0.80	0.44	0.160	20.80	0.26	0.97
0.27	10.0	172	68,000	0.75	0.80	0.44	0.160	32.50	0.26	0.96
0.34	4.0	330	8,399	0.26	0.29	0.27	0.160	1.31	0.29	0.86
0.37	8.0	215	50,000	0.50	0.56	0.31	0.183	15.60	0.28	1.03
0.41	4.0	340	9,983	0.28	0.28	0.25	0.184	1.12	0.39	0.74
0.45	3.0	450	5,800	0.25	0.29	0.26	0.193	0.55	0.49	0.62
0.45	3.0	910	12,800	1.00	1.20	0.26	0.209	6.10	0.39	0.83
0.45	4.0	353	11,100	0.25	0.30	0.25	0.198	1.38	0.38	0.82
0.45	4.0	500	15,650	0.50	0.60	0.24	0.196	3.90	0.38	0.81
0.45	4.0	612	19,700	0.75	0.90	0.24	0.202	7.40	0.38	0.85
0.45	4.0	705	22,800	1.00	1.20	0.25	0.203	11.20	0.38	0.83
0.45	6.0	233	24,700	0.25	0.30	0.25	0.200	3.00	0.39	0.81
0.45	6.0	405	43,500	0.75	0.90	0.25	0.202	16.10	0.38	0.83
0.45	8.0	175	43,800	0.25	0.30	0.25	0.198	5.40	0.38	0.82
0.45	8.0	250	62,000	0.50	0.59	0.24	0.197	15.30	0.38	0.82
0.45	8.0	305	78,000	0.75	0.90	0.25	0.203	30.00	0.37	0.88
0.45	8.0	350	90,000	1.00	1.20	0.25	0.203	45.00	0.38	0.85
0.45	10.0	140	67,500	0.25	0.30	0.25	0.196	8.50	0.375	0.83
0.45	10.0	280	140,000	1.00	1.20	0.25	0.203	70.00	0.38	0.84
0.50	4.0	250	10,000	0.125	0.16	0.26	0.250	0.40	0.63	0.63
0.50	4.0	350	14,000	0.25	0.33	0.27	0.250	1.00	0.70	0.55
0.50	4.0	424	12,550	0.25	0.31	0.174	0.186	0.97	0.634	0.46
0.50	4.0	598	17,770	0.50	0.62	0.174	0.187	2.73	0.64	0.45
0.50	4.0	600	24,000	0.75	0.93	0.27	0.250	6.00	0.62	0.64
0.50	4.0	731	21,700	0.75	0.93	0.176	0.186	5.00	0.638	0.46
0.50	4.0	775	31,000	1.25	1.64	0.28	0.255	12.00	0.67	0.59
0.50	4.0	946	28,000	1.25	1.55	0.176	0.186	10.75	0.638	0.46
0.50	6.0	282	27,800	0.25	0.31	0.18	0.185	2.14	0.64	0.46
0.50	6.0	488	48,150	0.75	0.93	0.18	0.186	11.10	0.64	0.46
0.50	6.0	630	62,250	1.25	1.55	0.18	0.186	23.90	0.64	0.46
0.50	8.0	125	36,000	0.125	0.16	0.26	0.225	1.40	0.64	0.55
0.50	8.0	176	50,000	0.25	0.31	0.26	0.227	4.00	0.61	0.59
0.50	8.0	211	49,000	0.25	0.31	0.18	0.186	3.76	0.64	0.46
0.50	8.0	305	86,000	0.75	0.93	0.26	0.225	20.00	0.63	0.56
0.50	8.0	366	85,000	0.75	0.93	0.18	0.186	19.60	0.64	0.46
0.50	8.0	395	110,000	1.25	1.55	0.26	0.225	42.00	0.64	0.55
0.50	8.0	473	109,820	1.25	1.55	0.18	0.186	42.10	0.64	0.46
0.60	1.5	1,150	2,300	0.25	0.36	0.20	0.240	0.36	0.36	1.04
0.60	4.0	430	16,300	0.25	0.36	0.20	0.240	2.56	0.36	1.05
0.60	8.0	215	65,000	0.25	0.36	0.20	0.240	10.20	0.36	1.04
0.60	10.0	172	102,000	0.25	0.36	0.20	0.240	16.00	0.36	1.05
0.75	4.0	598	21,400		0.31	0.088	0.224	2.03	0.65	0.69
1.00	1.5	1,150	3,750	0	0.28	0.16	0.386	0.56	0.30	2.10
1.00	3.0	232	4,500	0	0.025	0.08	0.286	0.06	0.30	1.50
1.00	3.0	465	9,000	0	0.10	0.08	0.286	0.45	0.31	1.40
1.00	4.0	173	8,000	0	0.025	0.086	0.290	0.10	0.30	1.50
1.00	4.0	340	20,372	0	0.16	0.14	0.380	0.76?	0.67?	0.88?
1.00	4.0	345	16,000	0	0.10	0.086	0.290	0.80	0.31	1.50
1.00	4.0	430	26,500	0	0.28	0.16	0.386	4.00	0.29	2.10
1.00	4.0	520	24,000	0	0.22	0.083	0.290	2.70	0.31	1.50
1.00	4.0	598	21,900	0	0.19	0.054	0.230	1.65	0.43	1.44
1.00	6.0	228	35,000	0	0.10	0.09	0.290	1.75	0.31	1.40
1.00	8.0	85	30,500	0	0.023	0.082	0.282	0.40	0.28	1.60
1.00	8.0	175	61,000	0	0.09	0.075	0.275	3.00	0.29	1.50
1.00	8.0	215	105,000	0	0.28	0.16	0.386	15.90	0.29	2.10
1.00	8.0	260	92,000	0	0.20	0.075	0.280	10.25	0.28	1.60
1.00	8.0	305	110,000	0	0.18	0.05	0.290	14.30	0.31	2.00
1.00	10.0	70	47,000	0	0.022	0.073	0.274	0.60	0.27	1.60
1.00	10.0	140	95,000	0	0.09	0.075	0.275	4.75	0.285	1.50
1.00	10.0	172	165,000	0	0.28	0.16	0.386	25.00	0.29	2.10

h_0 = feet of air corresponding to observed water gage when fan acts as a blower.

u = peripheral velocity, feet per second, $= 0.0524DN$;

o = area resistance orifice, square feet;

r = orifice ratio $= o/A$,

u.hp. and b.hp. = useful and brake horsepower,

E = efficiency $= \text{u.hp.}/\text{b.hp.}$,

k = variable pressure coefficient,

K and a = type constants defining k ,

f = type constant defining power,

g = acceleration of gravity $= 32.2$,

d = density of air, pounds per cubic foot, $= 0.075$ approximately.

Theoretical maximum head or "peripheral velocity pressure" is $u^2/2g$. The actual pressure, being less than this, is

$$H = \frac{ku^2}{2g} = h_0 + \frac{v^2}{2g} \quad [1]$$

Where H and h_0 are measured by gage as in exhaust and blowing experiments respectively. Making K the value of k when r equals zero, and assuming a straight-line graph for the total pressure

$$H = \frac{Ku^2}{2g}(1 - ar) \quad [2]$$

The velocity through the resistance orifice, omitting the *vena contracta* coefficient, is $\sqrt{2gH}$. Since the ratio of this velocity to the velocity in the tube is inversely as orifice ratio

$$v = r\sqrt{2gH} = ru\sqrt{K(1 - ar)} \quad [3]$$

and the volume delivered is

$$q = o\sqrt{2gH} = Aru\sqrt{K(1 - ar)} \quad [4]$$

Values of K range from 0.25 to 0.50, and of a from 0.65 to 0.85. Average values are $K = 0.38$, $a = 0.75$.

The useful work done by the fan (including velocity) is

$$\text{u.hp.} = \frac{qHd}{550} = 0.000158WQ \quad [5]$$

Brake horsepower, although approximately proportional to pressure, area and speed, tends to increase with the orifice ratio when these factors are constant. Formula 6 approximates a wide range of data.

$$\text{b.hp.} = f(1 + r^3)WND^3 \quad [6]$$

Values of f range from 0.00010 to 0.00025, averaging 0.00017.

Efficiency, found by dividing equation 5 by equation 6, is

$$E = \frac{0.000158Q}{f(1+r^3)ND^3} \quad [7]$$

When $f = 0.00017$ this equation reduces to

$$E = \frac{0.93Q}{(1+r^3)ND^3} \quad [7a]$$

In order to ascertain the relation existing between the efficiency and the orifice ratio; substitute in equation 7 the value of N derived from equation 2 and the value of Q derived from equation 4. These values are

$$N = \frac{60}{\pi D} \sqrt{\frac{10.4gW}{dK(1-ar)}} \quad [8]$$

$$Q = 15\pi D^2 r \sqrt{\frac{10.4gW}{d}} \quad [9]$$

Whence

$$E = \frac{0.000158\pi^2 \sqrt{K}}{4f} \times \frac{r\sqrt{1-ar}}{(1+r^3)} \quad [10]$$

Hence efficiency depends wholly on the fan constants and the orifice ratio and is independent of both size and speed. Putting $f = 0.00017$, $K = 0.38$ and $a = 0.75$.

$$E = \frac{1.41r\sqrt{1-0.75r}}{(1+r^3)} \quad [10a]$$

TABLE 2.—*Comparison of Results with Formulas*

$r =$		0.	0.250	0.270	0.450	0.500	0.600	0.750	1.000
$2gH/u^2 =$	Maximum....	0.430	0.280	0.440	0.260	0.280	0.200		0.160
	Minimum....	0.310	0.270	0.440	0.240	0.174	0.200	0.088	0.050
	Formula.....	0.380	0.309	0.303	0.252	0.238	0.209	0.166	0.095
$v/u =$	Maximum....		0.170	0.160	0.209	0.255	0.240		0.386
	Minimum..		0.106	0.160	0.193	0.185	0.240	0.224	0.230
	Formula.....		0.139	0.148	0.226	0.243	0.274	0.306	0.308
Efficiency,	Maximum...		0.540	0.260	0.490	0.700	0.360	0.650	0.43
	Minimum....		0.365	0.260	0.370	0.610	0.360		0.270
	Formula.....		0.312	0.333	0.473	0.495	0.516	0.492	0.352
$\frac{\text{b.hp.} \times 10^4}{60uA W} =$	Maximum....		0.680	0.970	0.880	0.640	1.050		2.100
	Minimum....		0.410	0.390	0.950	0.620	0.450	1.040	0.690
	Formula.....		0.688	0.699	0.702	0.751	0.774	0.837	0.978

COMPARISON OF FORMULAS WITH DATA

In Table 2, selected high and low values, taken from Table 1, are compared with the results of the formulas using average values for the

constants. The formulas reduced to the form required to make the comparison are:

$$\begin{aligned}\frac{2qH}{u^2} &= 0.38(1 - 0.75r) \\ \frac{v}{u} &= r\sqrt{0.38(1 - 0.75r)} \\ E &= \frac{1.41r\sqrt{1 - 0.75r}}{(1 + r^3)} \\ \frac{\text{b.hp.} \times 10^4}{60uH} &= 0.688(1 + r^3)\end{aligned}$$

PROPER SIZE OF FAN

The proper size of fan for stated volume and pressure, considered from a theoretical standpoint, is that size which will give the greatest mechanical efficiency. In practice this principle must be modified because it implies too large and costly a plant for low pressures. The scheme given below indicates the most efficient size up to a point where size is limited by the usual engineering practice.

Propeller fans usually operate between one-quarter orifice and no resistance, or wide open; that is between $r = 0.25$ and $r = 1.00$. The average efficiencies corresponding to these ratios, as given by equation 10a, are 31 and 35 per cent. respectively, but between these ratios efficiency attains a maximum. The ratio corresponding to maximum efficiency may be found by differentiating equation 10. The value of r so found satisfies the equation

$$1.5ar(1 - r^3) = (1 - 2r^3) \quad [11]$$

Substituting 0.75 for a , the value of r is found to be 0.617; and the maximum efficiency, corresponding to this value, found from equation 10a, is $E = 0.5164$. These figures harmonize with experimental results.

The most efficient size for given volume and pressure may be found by substituting the most efficient value of r in equation 9. Putting $r = 0.62$, solving for D , and reducing,

$$D = 0.023 \sqrt{\frac{Q}{W}} \quad [12]$$

Since this formula gives unnecessarily large fans for low pressures, engineering practice sets an arbitrary limit to diameter, depending on volume alone. This limit varies with local conditions between 0.024 and 0.040 multiplied by \sqrt{Q} . The average is

$$D = 0.032\sqrt{Q} \quad [12a]$$

Since equations 12 and 12a correspond when the water gage, W , equals 0.25 in. the best practice is to determine diameter by equation 12 when the pressure is above that figure; and by 12a when it is below.

The above rule may be varied to suit local conditions; but, to avoid waste of power, the orifice ratio should preferably not be less than 0.25; and, if stock fans are used, the speed limits given by the manufacturers should not be exceeded. With average values of the fan constants, the peripheral velocity implied by equation 12 will not exceed 12,000 ft. per min., provided the value of W is less than 1.8 in. The application of equation 12a to pressures below $W = 0.25$ in. implies an orifice ratio greater than $r = 0.62$, and a peripheral speed less than about 4500 ft. per min. When $r = 0.25$

$$D = 0.036 \sqrt{\frac{Q}{\sqrt{W}}} \quad [12b]$$

which permits a water gage of $W = 2.7$ in. without exceeding a peripheral speed of 12,000 ft. per min. To find the value of r corresponding to the smallest diameter consistent with a given speed limit, consider the following equation, which can be derived from equation 4:

$$\frac{Q}{D^2 r} = 47.12u \sqrt{K(1 - ar)} = 3148 \sqrt{W} \quad [12c]$$

Solving for r gives

$$r = \left[\frac{1}{a} - \frac{4463W}{aKu^2} \right] \quad [12d]$$

whence the required diameter can be found.

EXAMPLE OF COMPUTATION

Required the size of fan, speed and horsepower, to deliver 100,000 cu. ft. of air per min. at 1-in. water gage, fan installed as an exhausting plant. Also for 2-in. water gage, with a peripheral speed not exceeding 12,000 ft. per min. Since the gage is 1 in., diameter may be arranged to attain maximum efficiency without excessive speed. Using equation 12, $D = 0.023 \sqrt{100,000} = 7.3$ ft. The required speed is found from equation 2, which can be reduced to $u = \sqrt{4463W \div K(1 - ar)}$. Making $K = 0.38$, $a = 0.75$ and $r = 0.62$; u is found to be 148.5 ft. per sec., or 8910 ft. per min.; which corresponds to 390 r.p.m. Brake horsepower required, from equation 6, is $f(1 + r^3)WND^3 = 0.00017 \times 1.24 \times 1 \times 390 \times 389 = 32$ brake horsepower.

If 2-in. water gage had been required in the above instance, the most efficient diameter could not have been used without exceeding a peripheral velocity of 12,000 ft. per min. The largest advisable diameter, obtained by substituting $W = 2$ in equation 12b, is 9.5 ft. But as this diameter permits 2.7 in. water gage without exceeding 12,000 ft., a smaller and more efficient installation can be made. To find the smallest diameter consistent with this speed limit, put $u = 200$, $W = 2$, and the average values of K and a , in equation 12d. Then r is found to equal 0.55. Substituting this value in equation 12c, gives $D = 6.3$ feet.

ACKNOWLEDGMENTS

The author takes this opportunity to thank The American Blower Co. (Ventura Fan), The Jeffrey Manufacturing Co. (Straitflo Fan), The Buckeye Blower Co., and The New York Blower Co. for generously furnishing data for this study; also the Coppus Engineering Corpn. for data on a very interesting and unusual type of propeller fan, which has not been included merely because it differed from the subject type.

No comparison of the general merits of the several fans listed in the table has been intended. Construction, desirable from many points of view, is not always compatible with high mechanical efficiency; and he who wishes to purchase a fan must weigh all its advantages and disadvantages in the light of his own especial conditions.

DISCUSSION

G. E. McELROY, Pittsburgh, Pa. (written discussion).—This paper is an ingenious piece of mathematics that averages practical fan performance and data, but without very close agreement with any one set of data. Its usefulness, therefore, would seem to be limited to very approximate solutions of problems in fan selection in the absence of detailed data such as the fan manufacturers provide. Fan selection should be handled, I think, from curves or tables of generalized performance data where such a method is possible.

M. D. COOPER, Pittsburgh, Pa.—A practical example of the use of disk fans occurred a few months ago. We had occasion, in Masontown, Pa., to shut down a mine which was ventilated by a Capell type of fan of about 18 ft. dia. and 7 ft. wide. It was giving about 100,000 cu. ft. of air per min. The payroll at that time amounted to about 300 men. It now amounts to about 12 men, including the superintendent and the clerk. We had purchased power available, so we closed up the Capell fan and inserted in the side of the fan house a 6-ft. propeller-type fan operated by purchased power. That fan is running satisfactorily, and instead of keeping the boiler plant running and the power house and the steam-driven fan, we are now ventilating the mine with 40,000 cu. ft. of air per min. from the little fan, thereby saving a great deal in cost.

G. S. RICE, Washington, D. C.—What is the water gage?

M. D. COOPER.—About an inch. We are saving in power and in labor, and the mine is absolutely clear. Frequent examinations show that although there is a certain tendency for generation of black damp we are getting satisfactory results with this emergency arrangement.

G. S. RICE.—South African mining magazines¹ have reported the use of airplane propellers, of which there are three or four in series. They claim an efficiency as high as 70 per cent. That is surprisingly high, because I would have expected that the air would be made turbulent by the first propellers of the series, which would be unlike the case of airplane propellers that are cutting into a fresh body of air all the time.

I think the usual difficulty has been in connection with getting sufficient head in slippage which occurs when there is hard resistance.

¹ H. Briggs: The Ventilation of Mines Considered from the Engineering Standpoint—The Air Screw Fair. *Colliery Engrg.* (1928) 5, 317.

H. P. GREENWALD, Pittsburgh, Pa.—Mr. McElroy is compiling the results of a number of tests made in the metal mines at Butte, Mont., at the time of the year when temperatures change very rapidly. Natural ventilation in those deep shafts changes with temperature, and he finds that it had a most surprising effect on the total quantity of air. I believe he said there were times when as much as 60 per cent. of the total volume of air moving was induced by natural ventilation. With the fan shut down half of the original volume of air continued to circulate. Natural ventilation has been the subject of experimentation and papers² on it have been published in England. It has been found quite an important factor in their deep coal mines, particularly where the mine was warm and the outside air cold.

G. S. RICE.—In deep mines the natural ventilation is sometimes very strong, but you cannot rely on it always continuing so or in the same direction.

C. W. GIBBS, Harwick, Pa.—I have rather an interesting problem before me now in our mine. We have an auxiliary fan, steam-driven, and want to do away with the boiler system and the old type of steam fan. It is a question of whether we will install a high-speed fan of smaller diameter, and if so, what type of motive power will be used. We have electricity but we prefer to have an auxiliary fan independently driven. What information is there on the use of oil or gas Diesel engines in operating the fans, and what is the tendency to the high-speed fan as against those of larger diameter?

D. HARRINGTON, Washington, D. C.—As far as the second installation is concerned, there are a number of second installations in the country that are using either gas or oil. The Woodward Iron Co. in Alabama has one of its mines equipped with an auxiliary fan, which, by the way, undoubtedly saved the lives of a considerable number of men after an explosion about three years ago. The ordinary fan went out of order immediately, but in 15 min. the second fan was in operation and pulling normally.

The Empire Mining Co. in Southern Colorado has a similar installation. As far as I know, it is entirely satisfactory. They use it only for emergencies, possibly not more than half a dozen days a year, but it is in place and ready to go.

G. S. RICE.—What is the experience in the Pennsylvania coal mines with duplicate fans?

H. N. EAVENSON, Pittsburgh, Pa.—The mines with which I am connected have no installations other than steam and electric-driven fans. Steam-driven fans at gaseous mines are maintained in regular operation. The duplicate fans are electric. Sometimes during periods of shutdown, the electric-driven fan is operated constantly and the steam fan shut down, but when the mines are operating, the steam fans are generally depended upon. There are no installations at these plants driven by gas engine.

G. S. RICE.—I think there are quite a number of duplicate fans in the country, that use a different power circuit from the main ventilating fan.

H. N. EAVENSON.—We have always endeavored to have two sources of supply for electric-driven fans, using our own current normally.

J. A. GARCIA, Chicago, Ill.—I would like to call attention to the texrope drive for fans. This drive is merely a series of small V-shaped, composition ropes, each endless, operating in grooves on the pulleys.

² R. Clive: The True Effect of Natural Ventilation in Deep Mines. *Trans. Inst. Min. Engr.* (1923-24) 67, 268-315.

Our experience has been quite satisfactory with this improved rope drive, and we are now considering its use on fan drives as high as 500 hp. It permits the motor to be quite close to the fan, and it is not necessary to have long centers to get contact.

At a recent installation where a very large motor was used on the fan, the belt renewals and repairs cost close to \$2000 a year and with the multiple V-rope system, the repairs so far have been negligible.

We also have used this type of drive on screens satisfactorily.

We respectfully suggest that the Committee investigate this type of drive for fan service.

G. S. RICE.—In 1923, I saw such rope drives for fans in Europe, and they were spoken of very favorably then.

J. GARCIA.—This is not a rope drive in that a tension take-off is needed.

G. S. RICE.—That is true. It is an excellent point to bring out in order that this Committee may study the advantage in that method of driving a fan.

Recirculation of Air and Mine Gas Caused by Auxiliary Fans as Used in Coal Mines*

BY H. P. GREENWALD† AND H. C. HOWARTE,‡ PITTSBURGH, PA.

With an Introduction by G. S. RICE§

(New York Meeting, February, 1928)

INTRODUCTION

THE rapidly increasing use in coal mines of portable auxiliary fans, which are generally "blowers" employed in connection with canvas tubing, raises questions concerning the hazards of such equipment, especially where the practices are opposed to established methods of ventilating face workings. Half a dozen or more coal-mine explosions and some fires have been attributed to the use of auxiliary fans; moreover, competent mining engineers have found many dangerous installations and methods of use—for example, intermittent running of the fans in gassy mines with consequent accumulations of firedamp at the faces. These facts have led the U. S. Bureau of Mines to issue this recommendation:

"In the interest of safety, the Bureau of Mines, Department of Commerce, recommends that auxiliary fans or blowers should not be used in coal mines as a substitute for methods of regular and continuous coursing of the air to every face in the mine."

The comparatively recent introduction of new types of mining machines, mechanical loaders, face conveyors, and slabbing machines has encouraged the trial of new systems of face mining, all of which have for their object the rapid advance of the faces. Although the latter has certain economic advantages, in gassy mines the rate at which gas is given off is generally found to increase in proportion to the speed of advance into new territory, which in turn increases the difficulties of efficient ventilation of the face workings. Some of the systems of advance mining being tried in connection with new mechanizing methods practically compel the use of blower fans and tubing, as single narrow places are now driven 300 ft. or more without crosscuts. In some instances a series of parallel headings are driven without crosscuts pre-

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paratory to slabbing; reliance is then placed on auxiliary fans for ventilation, the return of one becoming the intake of the next. Whether some of these systems are necessary for efficient mechanization of mines is doubtful. At all events, they introduce new mining dangers.

The hazards of using auxiliary fans are well set forth in the paper¹ by H. I. Smith, given last year before the Mine Ventilation Committee of the Institute, and in the discussion that followed by Daniel Harrington and others.

This paper does not propose to discuss further the general question of the advisability of the use of auxiliary fans or the several hazards connected therewith, except the single one of accumulation of mine gas caused by recirculation. The chief object of the study was to devise means to minimize recirculation where auxiliary fans and tubing are used. The data on this subject were obtained by a series of tests made in the Bureau of Mines experimental mine, near Bruceton, Pa., by H. P. Greenwald and H. C. Howarth. The giving off of flammable gas at the face of an entry, as when an undercutting machine strikes a gas "feeder" or "blower," was simulated by piping natural gas to the face and liberating it there at a metered rate. The volume of the pure air current from which the auxiliary fan drew its supply and the volume of return air and gas were each carefully measured and samples of the latter mixture were taken and analyzed.

The first tests were made in a side entry 90 ft. long at right angles to the main entry in which the auxiliary fan was placed. Similar tests, using two fans, were made in the headings of the main entry and parallel air course, each of which extended 50 ft. in by an open crosscut. In the first group of tests the fan was placed successively in the mouth of the side entry and in the main entry at various specified distances out by the rib corner of the side entry, the tubing in the latter tests making a right-angle bend. It was found that the fan inlet should be in the main entry not less than 8 ft. on the intake side of the side entry, and in this location the fan could take only 40 per cent. of the air current in which it was placed. Increasing the distance to 16 ft. increased the available fraction of the main air current about 10 per cent. The effect of hanging check curtains to prevent recirculation was determined for one position of the fan, but the results, though slightly favorable, were not decisive enough to permit positive recommendation for their use. Under ordinary conditions, placing the fan further out by would be a better plan.

In the second group of tests, two fans were placed in the main entry out by the last crosscut. The tubing leading to the face of the main entry was without bends, while that leading to the face of the air course had two right-angle bends. The results were not found to differ appreci-

¹ H. I. Smith: The Use and Dangers of Booster and Auxiliary Fans, as Applied to Coal Mine Ventilation. *Trans.* (1927) 75, 629.

ably from those obtained in the side entry for the same ratio of fan delivery to main air-current volume.

Under the conditions of test, when the volume circulated by the auxiliary fan was approximately equal to that of the main current there was certain to be recirculation to greater or less extent, depending on the position of the fan. The maximum found was 50 per cent. when the fan was directly at the mouth of the heading. In one example, when the volume of gas liberated at the face was 1 per cent. of the volume handled by the fan, the percentage of gas in the return of the heading rose to 1.8 per cent. in $\frac{1}{2}$ hr. On the same basis, if the ratio of pure gas was 2.5 per cent. of the fan delivery, the percentage of gas in the return would rise to 4.5 per cent. In such case, there would probably be an explosive mixture somewhere between the face and a point opposite the discharge end of the canvas tubing.

Tests were made to determine how far the end of the tubing might be from the face and yet obtain a good stream of air at the face. It was found, under the conditions of test, that 35 ft. was the maximum distance advisable to obtain good mixing of the air and gas. This result was obtained in an unobstructed heading and does not indicate what concentration of gas might take place at the face if the flow of air from the end of the canvas tubing met obstructions, such as a full car standing near the face. To meet this condition, it seems advisable that the tubing should extend at least to a point opposite and to one side of where a car might be placed. The effect on recirculation of the frequent moving of cars in the main entry was not obtained, but as such effect might be important, it emphasizes the need of keeping the fan at a considerable distance outby the mouth of a heading and of having the main current at least $2\frac{1}{2}$ times the volume taken by the auxiliary fan.

A formula was developed for calculating the rate at which gas increased in the return for different conditions of recirculation and the maximum percentage of gas attained when the rates of gas liberation and delivery by the fan are varied.

It is believed that the data gathered by Greenwald and Howarth constitute a valuable contribution, covering certain phases but by no means all, of a timely subject which will better enable coal-mining officials to determine the degree of hazard in using auxiliary fans and tubing for face ventilation and when such equipment is used to determine the least hazardous arrangement.

ARRANGEMENT OF TESTS

The investigation was limited to determination of the precautions that must be taken to prevent recirculation during normal operation of the fan. Fig. 1 is a plan of the portion of the experimental mine in which the tests were conducted. No. 1 right is a single entry 100 ft. long, driven off

the main entry at a point 775 ft. from the pit mouth and 535 ft. from the face. Side openings from No. 1 right were bratticed off, making it in effect an unbroken dead end entry. The main ventilating current passed the mouth of No. 1 right in a straight line. A different arrangement was present at the face of the main entry where the main ventilating current made a right-angle bend through a crosscut in passing the entry dead end.

Fig. 2 is a plan view of the arrangements at No. 1 right entry in detail and on a larger scale. The fan was located in different tests at points B_1, B_2, B_3 and B_4 . At point B_1 the inlet of the fan was 16 ft. outby the outer

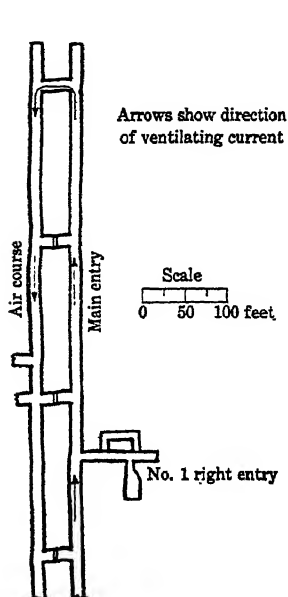


FIG. 1.—PLAN OF TEST ZONE IN EXPERIMENTAL MINE.

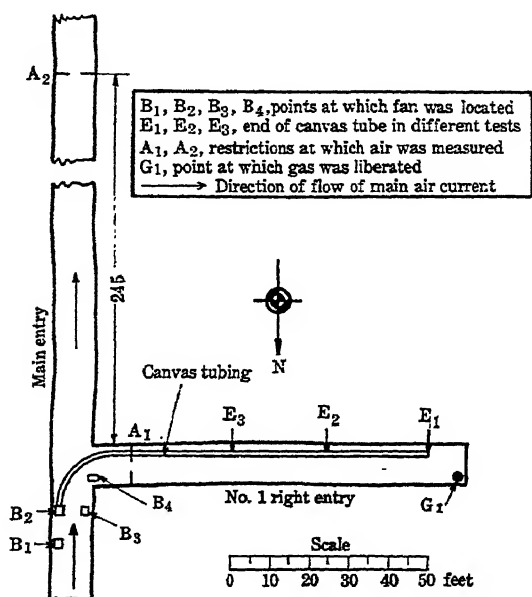


FIG. 2.—ARRANGEMENT OF TESTS IN NO. 1 RIGHT ENTRY.

rib line of No. 1 right entry; at points B_2 and B_3 it was 8 ft. outby this line. At point B_4 the fan was set in No. 1 right entry, but the inlet projected into the main entry 1 ft. beyond the rib line thereof. The tubing is shown as connected to the blower at location B_2 only; of course, readjustment was necessary when the blower was moved to the other locations.

In one group of tests the distance from the end of the canvas tubing to the face was varied, and the points where the end of the tube was located are designated by E_1, E_2 and E_3 . The corresponding distances from the end of the tube to the face are 10, 35 and 60 ft., respectively.

The volume of air delivered by the blower fan and the volume moving in the main air current were determined at points A_1 and A_2 respectively. In all tests, natural gas was liberated from a pipe at point G_1 , near the face of No. 1 right entry. The flow was very similar to that sometimes observed when a drill hole taps a pocket of firedamp.

Fig. 3 is a plan view showing the arrangement of the tests at the faces of the main entry and air course. Fans were placed at two locations, B_5 and B_6 and in each place the fan inlet was 8 ft. outby the outer rib line of the cut-through. The first tests were with a fan at location B_5 , ventilating the face of the main entry, a condition which is not shown in Fig. 3. The fan was then moved to location B_6 for a few tests, and finally a second fan was placed at B_5 with tubing arranged to ventilate the face of the air course. The air delivered by the fans was measured at restrictions

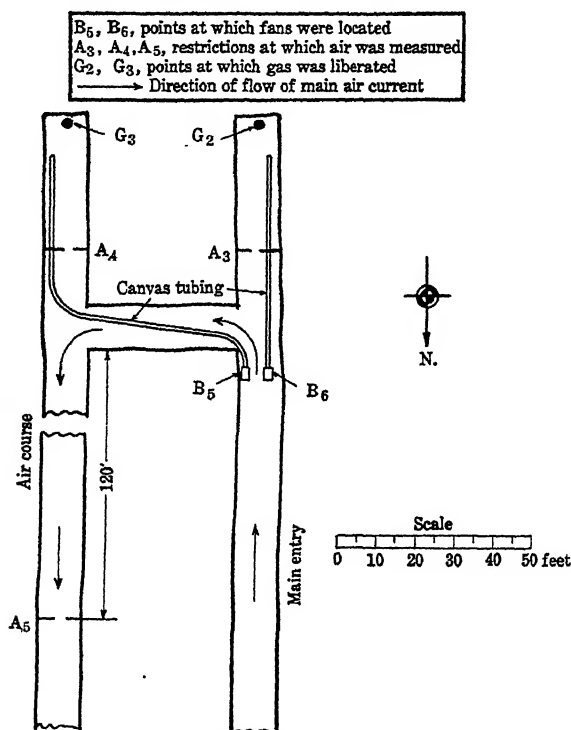


FIG. 3.—ARRANGEMENT OF TESTS AT FACE OF MAIN ENTRY.

A_3 and A_4 in the respective entries. The main ventilating current was measured at restriction A_5 in the air course. Gas was liberated from pipes at G_2 and G_3 at the faces of the respective entries as required for the different groups of tests.

FANS USED

Nearly all of the tests with one fan only were made with a blower fan driven by compressed air loaned for this work by the Coppus Engineering Co. Use of a motor-driven fan was not considered advisable in the first tests, as recirculation might at times produce an explosive gas-air mixture. The fan selected for the work had a rated delivery of 2700 cu. ft. per min. through 100 ft. of 12-in. pipe. The consumption of compressed air was given as 52 cu. ft. per min. of free air compressed to 75 lb. per sq. in.

The quantity of air entering the fan inlet is augmented slightly by the exhaust of the compressed-air drive, and more air delivered by the fan than enters the inlet. Fig. 4 shows this fan set at location B_4 (see Fig. 2) at the corner of No. 1 right entry. The first part of the test work showed that it was possible to control the experiments so that an explosive mixture would not be produced. Consequently it was possible under the controlled condition of tests to use an available motor-driven No. 2 Sirocco fan in the few experiments in which two fans were operated side by side.

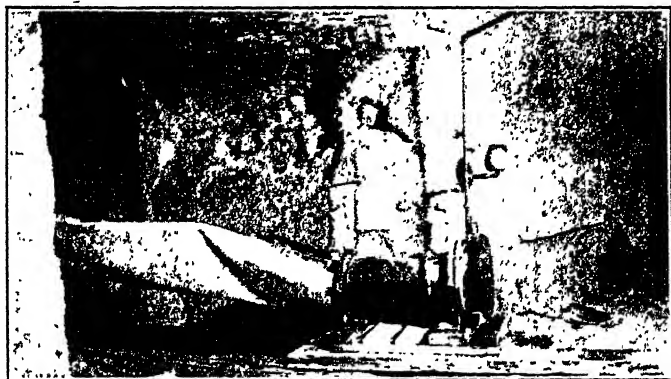


FIG. 4.—FAN AT CORNER OF NO. 1 RIGHT ENTRY, DRIVEN BY COMPRESSED AIR.

CONTROL AND MEASUREMENT OF AIR AND GAS FLOW

The main air current for these tests was induced by the mine fan, which has a capacity far in excess of the amount required for these tests. For accurate control, regulators were placed about 100 ft on the downstream side of air-measuring stations A_2 (Fig. 2) and A_5 (Fig. 3). These were of the slide-door type and were adjusted to give the desired volume immediately before each test.

Anemometers were judged sufficiently accurate for measuring the air movement. It is necessary, however, to have an air velocity of 200 ft. per min. or more to obtain good results with an ordinary anemometer, and this could be obtained for the present work only by restricting the size of the entry at the air-measuring plane.

At stations A_2 and A_5 (used only to measure the main ventilating current) the area was restricted to an opening exactly 2 ft. square. No sloped approach or departure surfaces were provided, which of course resulted in some turbulence and uneven flow of air through the restriction. This was compensated to some extent by dividing the opening into nine equal squares and taking an anemometer reading at the center of each square. The average of the velocities so obtained was taken as the average velocity over the entire area.

Much the same procedure was followed at stations A_1 , A_3 and A_4 where the air delivered by the fans was measured. It was feared, however, that such a restriction close to a corner might change the normal distribution of the air at that corner, consequently the restriction was so arranged that it could be opened to an area of 16 sq. ft. after the actual measurement was made. Careful tests were made to determine whether this change materially increased the volume of air delivered by the blower, and the conclusion was that it did not. Theoretically, the resistance of the fan pipe was 85 to 90 per cent. of the total for the circuit, hence a considerable change in the restriction would make little change in the total resistance. The characteristics of the fan would also enter into the problem, but it appears probable that the change in size of the restriction would probably change the volume not over 5 per cent. and might change it as little as 2 per cent.

The natural gas liberated at the faces was allowed to flow directly from a pipe. A flow of firedamp from a drill hole in the coal was thus simulated. The gas was passed through a meter, and suitable control valves were placed in the line. At the beginning of an experiment the flow was adjusted to approximate that desired and was subsequently unchanged. The average rate of flow for a test was taken as the total quantity admitted, divided by the elapsed time.

SAMPLING AND ANALYSIS OF GAS-AIR MIXTURES

The presence or absence of recirculation was determined primarily by the presence or absence of natural gas at the fan inlet, a logical proceeding, as the gas could come only from the face ventilated. Furthermore, the quantity of gas found at the inlet might be used as a measure of recirculation. It follows that the value of the work would be determined by the accuracy of sampling and analysis of the mixed gas and air entering the fan inlet. All samples were taken by water displacement and the natural gas content determined by combustion in an Orsatt apparatus. All determinations were made in duplicate and the average considered reliable to the nearest 0.1 per cent.

During most of the tests it was customary to take samples of the air passing out through the restriction in the dead end being ventilated and of the main return at a point 100 or more feet beyond the place ventilated. These and the sample at the fan inlet were taken as rapidly as a man could move from one point to the next and were in effect simultaneous samples.

TESTS WITH ONE FAN ONLY

Gas Concentration in Place Ventilated

Although determination of the presence or absence of recirculation centered at the fan inlet it was also important to ascertain the effect of recirculation on the concentration of natural gas in the place ventilated.

This matter was investigated in the first group of tests made in No. 1 right entry. It may appear at first that if the gas was being liberated continuously, even though at a small rate of flow, and a part of it was returned continuously by recirculation, the gas percentage in the dead end should rise steadily and in time an explosive mixture should always be obtained. This assumption is not correct, however, when the gas and air are being uniformly mixed, for the following reason: Gas is removed from the dead end in the return air, and the rate of outflow of the gas-air mixture is constant. However, as the percentage of gas in this mixture rises the outflow of pure gas from the dead end becomes larger and will finally balance the combined inflow from the gas feeder and the recirculated gas-air mixture. After this a constant percentage of gas should be found in the air in the place ventilated.

This theory can be developed mathematically, as shown in the Appendix of this paper. The method is not new, as it was used in a study of production of carbon monoxide from house heaters.² Other things being equal, the actual percentage of gas in the place ventilated will be directly affected by recirculation. Suppose a fan is delivering 1000 cu. ft. per min. of fresh air with no recirculation and gas is being liberated at the rate of 20 cu. ft. per min. If the air and gas are thoroughly mixed, the gas content in the air will be 2 per cent. Now if one-half of the air is recirculated, the gas content will rise to 4 per cent. If three-fourths of the air is recirculated the gas content will rise to 8 per cent., giving a highly explosive mixture.

The foregoing theory was subjected to experiment to determine its validity. Test No. 4 was made with conditions under which strong recirculation occurred. The fan was located at position B_2 (Fig. 2) and delivered 1010 cu. ft. of air per min. to the face of No. 1 right entry. The volume in the main air current was also 1010 cu. ft. per min. Gas was liberated at the face of No. 1 right at the rate of 10.4 cu. ft. per min. With no recirculation and uniform mixing, there should have been 1.0 per cent. gas in the air leaving No. 1 right entry. Samples of this return air were taken at position A_1 at intervals of $\frac{1}{2}$, $1\frac{1}{2}$, $2\frac{1}{2}$ and $4\frac{1}{4}$ hr. after the test was started. The analyses of these samples gave 1.6, 1.5, 1.6 and 1.7 per cent. natural gas respectively. This would indicate that 37 per cent. of the air was being recirculated. The gas content was essentially constant, as the theory predicted, and apparently less than $\frac{1}{2}$ hr. was required to obtain constant conditions in this test. The equation developed in the Appendix indicated that essentially constant conditions should be obtained in less than $\frac{1}{2}$ hr. and the primary rise in gas percent-

² G. W. Jones, L. B. Berger and W. F. Holbrook: Carbon Monoxide Hazards from House Heaters Burning Natural Gas. U. S. Bur. Mines *Tech. Paper* 337 (1923), 23 and 24.

age should be very rapid. To prove this, another test was made in which samples were taken at much closer intervals.

Test No. 20 was made with the fan located as in test 4 and delivering 1050 cu. ft. of air per min. The volume in the main air current was only 940 cu. ft. per min., and recirculation was consequently higher. Natural gas was liberated at the rate of 10.0 cu. ft. per min., which would give 0.95 per cent., in the return from No. 1 right if thoroughly mixed in the absence of recirculation. Samples were taken at position A_1 at intervals of 3, 6, 9, 12, 18, 30 and 60 min. after the start of the test. Analyses of these samples showed the gas content to be 0.5, 1.0, 1.2, 1.4, 1.6, 1.7 and 1.8 per cent., respectively. These figures indicate that approximately one-half of the air was recirculating. The gas content reached 1.0 per cent. in 6 min. and 1.6 per cent. in 18 min., and rose only an additional 0.2 per cent. in the next 42 min. The results indicated that no test need be conducted more than 30 min. to obtain essentially constant conditions.

Although the foregoing tests showed that a constant mixture could be obtained at one point, it was necessary to determine whether or not variations existed at different points in No. 1 right entry. This was done by taking samples as rapidly as possible at points in No. 1 right, 15, 30, 45, 60, 70 and 80 ft. from its mouth. These were all taken midway between roof and floor and the last was even with the end of the canvas tubing. Analyses of these samples gave gas contents of 1.8, 1.8, 1.8, 1.5, 1.9 and 2.0 per cent. There was certainly no variation in the first 45 ft. The low value at 60 ft. might have been due to a large eddy of fresh air from the fan pipe or it might have been an error. As no check test was made, the matter remains undecided. The two samples nearest the gas feeder show slight increases in gas concentrations. On the whole, however, the mixing of the gas and air was more uniform and thorough than would be expected. As no important differences in gas concentration were found along No. 1 right entry, it remained only to detect such as might exist vertically. Simultaneous samples were taken at the roof and center of the entry at two different points. The gas content in these samples differed only by 0.1 per cent. and it was concluded that no vertical stratification existed.

The foregoing tests were made with the end of the fan tube approximately 20 ft. from the face. It was evident that if a considerable distance separated the face of the entry and the end of the tube part of the air would be diverted to join the return without reaching the point to be ventilated. Tests were made to determine roughly the maximum distance that could be allowed. The end of the tube was successively 10, 35, and 60 ft. from the face of No. 1 right entry. Air samples were taken at points 15 and 80 ft. from the entry mouth. If uniform mixing of gas and air was obtained there should have been little difference in gas content at the two points. If part of the air was being diverted, the outer point

should have shown a lower gas content than the inner. When the end of the tube was 10 and 35 ft. from the face the samples taken at the two points differed in gas content by 0.1 per cent. When the distance was increased to 60 ft. the difference in gas content increased to 0.6 per cent. It was concluded that the end of the fan tube should be not more than 35 ft. from the face of the entry. In the subsequent experiments the distance varied from 10 to 25 ft., depending on the location of the blower.

EFFECT OF FAN POSITION ON RECIRCULATION

To avoid recirculation, the fan inlet must be in fresh air and the quantity of air entering it must be not greater than the quantity available in the main current. These are purely qualitative statements, and the present group of experiments was made to determine what proportion of the main ventilating current the auxiliary fan could take without causing recirculation when the fan was in a number of different positions. The first and most extensive set of tests was conducted at No. 1 right entry.

Tests at No. 1 Right Entry

Referring to Fig. 2, the fan was successively located at points B_1 , B_2 , B_3 and B_4 . At each of these locations the fan was operated to deliver approximately 1000 cu. ft. of air per min. to the face of No. 1 right. The air volume in the main ventilating current was set at approximately 1000 cu. ft. per min. for the first test at each location. It was then raised in increments of 1000 cu. ft. per min. until recirculation was no longer found, after which a test was made with an air volume intermediate between the last two in order to determine more closely the actual point at which recirculation ceased.

In all of these tests natural gas was liberated at the face of No. 1 right entry at the rate of 10 cu. ft. per min. approximately. After each test had been in progress $\frac{1}{2}$ hr., air samples were taken at the fan inlet, location A_1 in No. 1 right entry and location A_2 in the main entry. It was considered that the ratio of the air volume taken by the fan to that in the main ventilating current approaching it was the determining factor in recirculation rather than the absolute value of either. All the tests in which the fan delivered approximately 1000 cu. ft. per min. were in agreement with this conclusion, but in order to test the matter further the tests at location B_2 were repeated with the fan delivering 2000 cu. ft. per min. The conditions and results of the tests are given in Table 1.

An inspection of the table shows that when the fan took a volume equal to the main current, strong recirculation was present irrespective of the location of the fan. In test 10, at location B_1 gas could be detected near the roof 75 to 100 ft. on the fresh air side of the fan, showing that the recirculating current was passing back along the roof this distance before joining the fresh intake air which was traveling near the floor.

TABLE 1.—*Results of Tests with a Single Fan at No. 1 Right Entry*

Test No.	Air Delivered by Fan, Cu. Ft. per Min.	Air in Main Current, Cu. Ft. per Min.	Ratio of Fan Delivery to Main Current	Gas Liberated at Face, Cu. Ft. per Min.	Analyses of Samples, Per Cent. Natural Gas in Air at		
					Fan Inlet	Return of No. 1 Right	Main Return
Fan at Location B_1							
10	1020	1020	1.00	10.6	0.2	1.5	0.9
12	1020	1420	0.72	9.2	0.0	1.3	0.7
11	1020	2160	0.47	10.3	0.0	1.1	0.5
Fan at Location B_2							
7	1030	1000	1.03	9.8	0.8	1.7	0.7
8	1030	1880	0.55	10.3	0.1	1.3	0.7
13	1020	2530	0.40	9.0	0.0	0.9	0.4
9	1030	3020	0.34	10.0	0.0	1.2	0.5
54	2020	1960	1.03	17.3	0.6	1.3	1.0
53	2020	3030	0.67	18.8	0.2	1.0	0.5
52	2020	4040	0.50	19.5	0.1	1.0	0.5
55	2010	5000	0.40	23.0	0.0	0.9	0.5
Fan at Location B_3							
14	980	1020	0.96	9.9	0.8	1.6	0.8
15	980	1940	0.50	9.7	0.1	1.4	0.6
16	980	2470	0.40	10.0	0.0	1.2	0.6
Fan at Location B_4							
17	1050	1020	1.03	9.9	1.2	2.1	1.3
18	1050	1990	0.53	10.0	0.1	1.5	0.7
19	1050	2460	0.43	10.2	0.0	1.4	0.5

It was evidently not practical to set the fan sufficiently far from No. 1 right to prevent recirculation of the air under the conditions obtaining in this test. In test 11, recirculation was stopped when the volume in the main current was doubled and no gas was found at the fan inlet when an intermediate amount of air was circulated in the main current in test 12. Nevertheless, a higher percentage of gas was found in the return from No. 1 right entry in test 12 than should have been present had there been no recirculation. Either recirculation existed or there is an error in the analysis of the sample.

At location B_2 the fan was operated to deliver approximately 1000 cu. ft. per min. in one set of tests and 2000 cu. ft. per min. in another. In both cases the volume in the main current had to be increased to 2.5 times the fan delivery before recirculation ceased (tests 13 and 55). Exactly the same condition was found at location B_3 (test 16), and moving

the fan across the entry to the opposite rib apparently made no change in the amount of recirculation.

At location B_4 no gas was found at the fan inlet when the main air current was 2.5 times that delivered by the fan (test 19). However, the gas content of the return from No. 1 right was considerably higher than it should have been without recirculation, and an error of some kind is indicated. In any case, location B_4 is not a safe one for use in an operating mine, especially since mine-car movements would tend to deflect the return into the fan inlet. A motor-driven fan would be directly in the return air from the face, and the motor would be in position to ignite any explosive mixture that might accumulate from any cause in the place ventilated. As a fundamental safety precaution, any auxiliary fan should be placed on the fresh-air or intake side of a crosscut or side opening. Moreover, these tests indicate that when the distance on the intake side is 8 ft. the fan should not take more than 40 per cent. of the air current approaching it. When the distance on the intake side is 16 ft. the quantity taken can be increased to 50 or 55 per cent., but the tests did not determine this point exactly.

The gas analyses can be checked by computation from the test conditions. Thus, when constant conditions have been established the percentage of gas in the main return should be the rate of gas inflow at the face divided by the volume of air in the main return. Also, when there is no recirculation the percentage of gas in the return from No. 1 right entry should equal the rate of gas inflow divided by the air volume delivered by the fan. Values thus computed agree satisfactorily with the analyses except in tests 12 and 19, where the return from No. 1 right carried an abnormally high percentage of natural gas. Undetected recirculation is suspected.

Tests at Face of Main Entry

As noted before, the condition of flow of the main ventilating current is not the same for the location at No. 1 right entry and that at the face of the main entry. At the former place the main current continues past the side opening in a straight line, while at the latter it makes a right-angle bend. It was conjectured that the limiting conditions of recirculation might be different in the two cases, and tests were made to see if such a difference existed.

The fan was placed successively at locations B_5 and B_6 (see Fig. 3) and was used to ventilate the face of the main entry. The arrangement for location B_5 in the first group of tests does not appear in Fig. 3. At both locations the fan inlet was 8 ft. from the outby corner of the crosscut, a distance selected as similar to the distance of locations B_2 and B_3 from the outer rib corner of No. 1 right entry. The results of the tests made are given in Table 2.

TABLE 2.—*Results of Tests with a Single Fan at the Face of the Main Entry*

Test No.	Air Delivered by Fan, Cu. Ft. per Min.	Air in Main Current, Cu. Ft. per Min.	Ratio of Fan Delivery to Main Current	Gas Liberated at Face, Cu. Ft. per Min.	Analyses of Samples. Per Cent. Natural Gas in Air at		
					Fan Inlet	Return of Dead End	Main Return
Fan at Location B_5							
41	1000	1010	0.99	8.7	0.8	1.2	0.7
42	1000	1990	0.50	10.5	0.3	0.7 ^a	0.3 ^a
43	1000	2560	0.39	10.3	0.0	1.0	0.6
51	2070	4050	0.51	17.6	0.0	0.9	0.6
Fan at Location B_6							
47	990	1460	0.68	10.2	0.4	1.4	0.9
45	1040	2000	0.52	11.0	0.0	0.7 ^a	0.5
46	1040	2010	0.52	10.0	0.0	1.1	0.6
44	1040	2560	0.41	11.2	0.0	0.9	0.5

^a This analysis evidently in error.

When the fan was delivering 1000 cu. ft. per min. at location B_5 it was necessary for the main current to be 2.5 times the fan delivery to prevent recirculation (test 43). This is the result obtained at the similar location (B_2) outby No. 1 right entry. When the volumes of air in the main entry and fan delivery were both doubled there was apparently less recirculation. The difference can be noted when tests 42 and 51 are compared. Test 42 gave recirculation, whereas test 51 did not. The higher straight-line air velocity involving impact effects in the main entry might have changed the path of the eddy currents near the fan, but results of this nature were not obtained at No. 1 right entry, and further tests should be made before an application of the results of test 51 is attempted.

Location B_6 appeared to be slightly better than B_5 in preventing recirculation; thus in tests 45 and 46 the fan delivered approximately 1000 cu. ft. per min., and the air volume in the main entry was double that amount. No recirculation was found, whereas in test 42 with similar conditions at location B_5 recirculation was present. Confirming evidence is offered by tests 47 and 41. The differences should be made the subject of further investigation. For practical purposes one rule may be drawn covering installations at points B_2 , B_3 , B_5 and B_6 which will have a margin of safety for the last two locations. A fan placed at any one of these four points should take not more than 40 per cent. of the ventilating current approaching it; or, stated in another manner, the main current should be not less than 2.5 times the fan delivery.

The gas percentages found in the main return may be checked by computation, as in the tests at No. 1 right entry. Values so computed are in satisfactory agreement with the analyses, except for the main-return sample of test 42 and the dead end return sample of test 45 in which errors of analysis are indicated.

REDUCTION OF RECIRCULATION BY CHECK BRATTICES

The foregoing tests showed that recirculation was due to eddy currents which traveled near the roof and swept down to join the intake air at some point back of the fan. It was conjectured that these eddies might be stopped by check brattices placed either wholly or partly across the entry at some point between the fan inlet and the opening into the space

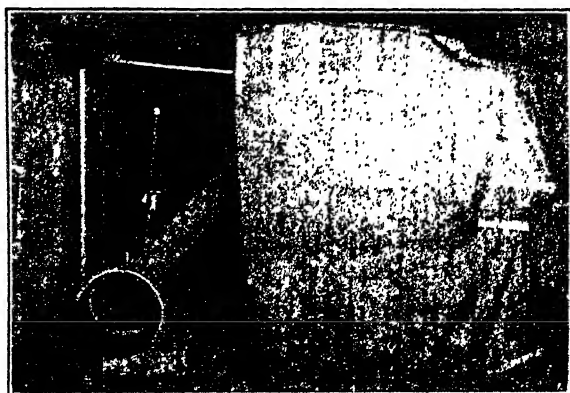


FIG. 5.—CHECK BRATTICE USED IN CONNECTION WITH FAN AT NO. 1 RIGHT ENTRY.

ventilated. In an operating mine such a brattice would frequently be installed at points on a trolley haulageway and could not be extended entirely across the entry.

In the tests the fan was located at point B_2 , close to the east rib of the main entry, with the tubing as shown in Fig. 2. The brattice was placed immediately inby the fan inlet, and in every case there was a clear space of 3 ft. between one edge of the canvas and the rib, such as would be necessary for passage of a trolley wire. The effect of changing this space from one side to the other of the entry was investigated. The principal variable, however, was the vertical length of the brattice. At the point where the brattice was erected the entry was 9 ft. 8 in. wide; 6 ft. 4 in. high at the sides and 7 ft. 4 in. high at the center. The air delivery by the fan was kept closely the same in all tests but three different volumes were used in the main current. Fig. 5 shows a brattice extending to the floor, with the 3-ft. opening on the same side of the entry as the fan. In order to speed up the work as much as possible air samples were taken at the fan inlet only, and the effect of the brattice on recirculation was judged

from the change in gas content at that point. The tests are divided into three groups, depending on the amount of air in the main ventilating current, and the results are given in Table 3.

TABLE 3.—*Results of Tests with Check Brattice*

Test No.	Air Delivered by Fan, Cu. Ft. per Min.	Air in Main Current, Cu. Ft. per Min.	Ratio of Fan Delivery to Main Current	Gas Liberated at Face, Cu. Ft. per Min.	Clear Space below Brattice, Feet	Opening at East or West Rib	Gas in Air, at Fan Inlet, Per Cent.
GROUP 1. TESTS WITH RATIO APPROXIMATELY 1.0							
7	1030	1000	1.03	9.8	"		0.8
26	980	960	1.02	10.5	4	West	0.4
27	980	960	1.02	10.5	2	West	0.5
28	980	960	1.02	10.5	0	West	0.4
29	980	960	1.02	10.5	2	East	0.3
31	1030	1000	1.03	11.5	0	East	0.2
GROUP 2. TESTS WITH RATIO APPROXIMATELY 0.7							
35	1020	1500	0.68	10.0	"		0.4
34	1020	1500	0.68	10.0	4	East	0.4
33	1020	1500	0.68	11.5	2	East	0.4
32	1020	1500	0.68	11.5	0	East	0.0
36	1020	1500	0.68	10.0	2	West	0.1
GROUP 3. TESTS WITH RATIO APPROXIMATELY 0.5							
8	1030	1880	0.55	10.3	"		0.1
38	1040	2020	0.51	10.6	4	West	0.1
37	1040	2020	0.51	10.6	2	West	0.1
40	1040	2020	0.51	10.6	2	East	0.1

" No brattice used in this test.

The tests of group 1 show that when the fan delivery was closely the same as the air volume in the main current there was strong recirculation with no brattice present (test 7). None of the brattices used stopped this recirculation completely although the amount was reduced greatly. Brattices extending to the floor were most effective and the least recirculation was obtained when the opening was on the opposite side of the entry from the fan (test 31).

In group 2 the original recirculation was but half that of group 1. Nevertheless, when the opening was on the same side as the fan it was necessary to extend the brattice to the floor before recirculation was stopped (test 32). When the opening was on the opposite side of the entry and there was a 2-ft. space between the floor and the bottom of the brattice, recirculation was still present (test 36), but the amount was much less than in the corresponding test with the opening at the east rib (test

33). A brattice extending to the floor would apparently be effective, irrespective of the location of the opening.

In group 3 there was only slight recirculation with no brattice present, but brattices extending to within 2 ft. of the floor failed to stop it. The results of group 2 proved that no recirculation would be found with brattices extending to the floor and the tests were not made.

Although the brattices reduced and in some cases stopped the recirculation, their value is too small to warrant their regular use. Moreover, there is always danger of their being damaged and having their small effectiveness wholly destroyed.

TESTS WITH TWO FANS OPERATING SIDE BY SIDE

When the faces of two parallel entries are ventilated by auxiliary fans the work may be accomplished in one of two ways. A single fan with a branched pipe may be used or two fans may be operated side by side. The first arrangement was not tested in the present work, as it does not appear that it should yield results essentially different from those obtained with a single line of tubing.

When two fans are operated side by side both must be placed in the entry which is acting as intake in so far as the particular location is concerned. If this is not done the second fan will operate in the return air from the first, a condition that would increase recirculation in the second dead end, and is not safe. If both are properly placed the arrangement is in general like that shown in Fig. 3. Three tests of this arrangement were made. Air measurements were made at points A_3 , A_4 and A_5 and air samples were taken at these points, the fan inlets, and the middle of the last cut-through. The fan inlets were 8 ft. out by the outer rib line of the cut-through. Data on the tests are given in Table 4.

In test 48, each fan was delivering approximately 1000 cu. ft. per min., and the combined delivery was closely the same as the volume in the main current. Recirculation was present but was much less than was found when the delivery of a single blower at location B_5 was equal to the main air-current volume. Thus in test 41 (Table 2) there was 0.8 per cent. gas at the fan inlet, as compared with 0.2 and 0.3 per cent. in test 48. In test 49 the absolute deliveries of the fans were slightly higher, but they were relatively unchanged. The main air current was increased in volume, so that the combined delivery of the two fans was but 70 per cent. of it. A trace of recirculation was found. In test 50, the delivery by the air-course fan was doubled, that of the entry fan was unchanged, and their combined delivery was closely the same proportion of the main current as in test 49. No gas was found at the fan inlets and there appeared to be no recirculation.

All three tests indicate that the two fans operating in parallel produced less recirculation than a single fan operating under the same relative

TABLE 4.—*Results of Tests with Two Fans Operating Side by Side*

AIR MEASUREMENTS							
Test No.	Air Delivered by Fan, Cu. Ft. per Min.			Air in Main Current, Cu. Ft. per Min.	Ratio of Fan Delivery to Main Current		
	Entry	Air Course	Total		Entry	Air Course	Total
48	990	1010	2000	1990	0.50	0.51	1.01
49	1060	1040	2100	2990	0.35	0.35	0.70
50	1000	2100	3100	4450	0.22	0.47	0.69

GAS MEASUREMENTS								
Test No.	Gas Liberated at Face, Cu. Ft. per Min.		Analyses of Samples. Per Cent. Natural Gas in Air at					
	Entry	Air Course	Entry Fan Inlet	Air Course Fan Inlet	Entry Return	Air Course Return	Middle of Cut-through	Main Return
48	11.2	8.1	0.3	0.2	1.6	1.2	1.1	1.1
49	9.0	10.3	0.1	0.1	1.0	0.9	0.6	0.7
50	8.4	9.7	0.0	0.0	0.6	0.5	0.4	0.5

conditions. The most plausible explanation of this is the fact that the incoming fresh air divides and enters a fan inlet on each side of the entry instead of being crowded to one side of the entry only, as it was when there was but one fan. Possibly a greater degree of recirculation would be found if both fans were on the same side of the entry, but proof of this must await further experiment. In the meantime, a measure of safety can be realized by applying a modification of the rule developed for a single fan; namely, that when the inlets of two fans operating side by side are 8 ft. out by the outer rib of the crosscut their combined delivery should be not more than 40 per cent. of the volume of the air current in which they are located.

CONCLUSIONS

The foregoing tests are by no means a complete investigation of the recirculation problem, but certain conclusions can be drawn from them which will promote safety in auxiliary-fan installations until they can be confirmed or modified by further experiment. This is without reference to dangers from accumulations of gas when the fans are stopped or to the potential hazard of gas ignition accompanying electric motors which may be used to drive them.

1. An auxiliary fan should always be installed on the fresh-air side of the place to be ventilated, and the fan inlet should be not less than 8 ft. from the nearer rib of the place ventilated or of the last cut-through, as the case may be.

2. When so placed the fan should take not more than 40 per cent. of the air current in which it is placed.

3. If the distance from the fan to the nearer corner be increased to, say, 16 ft., the danger of recirculation is decreased.

4. It is not practical to place a fan so that it may take the entire current in which it is placed.

5. The use of a check brattice to aid in avoiding recirculation is only justified in emergency.

6. When two fans are operating side by side, their inlets should be not less than 8 ft. from the nearer rib of the cut-through, and their combined delivery should be not more than 40 per cent. of the air current in which they are placed.

APPENDIX

Mathematical Consideration of Recirculation

Consider an arrangement such as is illustrated in Fig. 2 and assume that the gas and air are uniformly mixed. Use the following nomenclature:

A = decimal fraction of mixed gas and air leaving the dead end and passing to the main return.

B = volume handled by auxiliary fan, cubic feet per minute.

C = percentage of gas in mixture in dead end at any time.

C_o = percentage of gas in mixture in dead end at instant recirculation starts.

E = rate at which gas is carried out of the dead end, cubic feet per minute.

e = base of Napierian system of logarithms.

G = rate at which gas is liberated in the dead end, cubic feet per minute.

M = volume of gas in dead end at any time, cubic feet.

M_o = volume of gas in dead end at start of recirculation, cubic feet.

N = volume of air in main current, cubic feet per minute.

P = percentage of gas in main current.

R = rate at which gas enters inlet of fan, cubic feet per minute.

S = volume of air taken from main current by fan, cubic feet per minute.

t = time in minutes after start of recirculation.

V = volume of dead end, cubic feet.

From the nature of the case the following hold true:

$$\text{Fraction of air recirculated} = 1 - A$$

$$C = \frac{100M}{V}$$

$$E = \frac{BM}{V}$$

$$S = B - (1 - A)B = AB$$

$$R = \frac{(1 - A)BM}{V} + \frac{ABP}{100}$$

The value for R is the sum of the gas obtained from recirculation and from the main air current.

The following relation holds true: The rate at which the volume of pure gas increases in the dead end is equal to the rate at which gas is liberated in the dead end minus the rate at which gas is carried out of the dead end plus the rate at which gas enters the fan inlet. This may be expressed as a differential equation:

$$dM = Gdt - Edt + Rdt$$

This equation may be put in form for integration by substituting above values of E and R . Then dM is to be integrated from M_o to M and dt is to be integrated from o to t .

$$\int_{M_o}^M \frac{-\frac{AB}{V}dM}{G + \frac{ABP}{100} - \frac{ABM}{V}} = \int_o^t -\frac{AB}{V}dt$$

Integrating and reducing this equation with the aid of the equations $M = \frac{CV}{100}$ and $M_o = \frac{C_oV}{100}$ the final expression becomes

$$C = \left[\frac{100G}{AB} + P \right] \left[1 - \frac{1}{(e)^{\frac{ABt}{V}}} \right] + \frac{C_o}{(e)^{\frac{ABt}{V}}} \quad (1)$$

A study of this equation shows that time enters only as an exponent of e , and e is present only in the denominator of two fractions. It follows that as time increases indefinitely the values of these fractions approach zero. The second term of the right-hand side then vanishes, the second parenthesis of the first term becomes equal to unity and the equation reduces to

$$C = \frac{100G}{AB} + P. \quad (2)$$

This is the value that the gas content of the air in the dead end will approach under constant conditions and it should be noted that any gas present in the air approaching the auxiliary fan adds directly to the percentage found in the place ventilated. In the present tests $P = 0$ and the equation becomes

$$C = \frac{100G}{AB} \quad (3)$$

By definition $A = 1.0$ when there is no recirculation. The equation then is the simple form $C = 100 \frac{G}{B}$ used in computations in previous paragraphs; that is, in the absence of recirculation the gas percentage in the place ventilated is the rate at which pure gas is liberated divided by the rate of delivery by the fan. More strictly speaking, this should be

$C = 100 \frac{G}{B + G}$ but the error caused by omitting G from the denominator is much less than the error of experiment. As recirculation appears, A becomes less than one (equation 3) then C will become larger, hence the suspicion that a high value of C may be due to undiscovered recirculation. If one-half the air is recirculated the value of A becomes 0.50 and the value of C is doubled. Similarly if 75 per cent. of the air is recirculated $A = 0.25$ which multiplies the value of C by 4.

On page 172 reference is made to test 20, in which samples were taken at short intervals during the time that the gas percentage was increasing. The recirculation in this test may be determined from equation 3, in which $G = 10.0$, $B = 1050$, and $C = 1.8$ at the end of 60 min. Then $A = 0.53$ and the recirculation is $1 - A = 0.47$ or 47 per cent. Equation 1 may then be used to compute the value of C at the short intervals (3, 6, 9, 12, 18 and 30 min.) at which samples were taken. For this experiment $P = 0$, $C_o = 0$ and $V = 55 \pm 0$. The values so computed are compared with the analyses of the samples collected in Table 5.

TABLE 5.—*Comparison of Values Obtained by Computation and Analysis, Test No. 20*

Time, Minutes	Gas in Air Current	
	Computed, Per Cent.	Analyzed, Per Cent.
3	0.5	0.5
6	0.8	1.0
9	1.1	1.2
12	1.3	1.4
18	1.5	1.6
30	1.7	1.7

The values obtained by analysis are consistently higher than those obtained by computation, but the difference is too small to have any significance. In only one case (at 6 min.) does the difference reach 0.2 per cent. The agreement is better than would be expected in such large-scale experiments.

DISCUSSION

H. I. SMITH, Washington, D. C. (written discussion).—I am sure that these timely tests by the Bureau of Mines and the Bureau's promptness in publishing the results is greatly appreciated by those using or contemplating the use of auxiliary fans in coal mines. Industry cannot wait for delayed publications and if the results of tests by Government and research agencies are delayed or not made available, industry must forge ahead and learn its lessons by experience or by private research even though lives and property may be sacrificed in the experience. The number of lives that may be saved as a result of these tests in the experimental mine and the prompt publication of the data can never be estimated.

I am particularly interested in the results of these tests in view of the statement on page 260 of the 1927 *Transactions* of the National Safety Council: "If the fan is properly spaced the risk of recirculation does not exist."

In the introduction the authors refer to the rapid increase in the use of blower fans. I have not found this tendency on public and Indian lands under departmental lease, and I hope some one may give some information as to whether this increase is a general resort to blower fans in lieu of brattices or if the increase is due to a large number of such fans in a relatively small number of mines, or if their use is confined to particular coal fields, and I can hardly fully agree with the authors that the rate at which gas is given off is generally bound to increase in proportion to the speed or advancement into the new territory. I concede that other things being equal more methane per minute will come from entries being driven 20 ft. a day than one being driven 10 ft., and that greater provision must be made to handle gas at the face of the entry 300 ft. ahead of the last breakthrough than an entry with break-throughs at intervals of 65 ft., but the amount of gas per ton of coal will be less for the rapid entry work or in the case of concentrated work following development, with the result that the total amount of gas from any mine using concentrated mining methods will be less than where the mining is scattered. Moreover, the air can be concentrated as well as the mining operations and the gas hazard should be reduced with care, but greater care will be required in handling the ventilation to assure that there will not be interruptions or too rapid movement of accumulations of methane.

The authors have apologized for the analyses of samples taken. I do not believe that apologies are necessary and I am rather inclined to believe that the analyses are correct, for it is not expected that methane will always be thoroughly diffused within such short distances from the place of its exudation. The sample represents but a very small portion of the section and the tailing out of gas could easily cause a variation in the amount of methane that is obtained in the vacuum tube. Methane and air diffuse rather slowly in mines. At one place it is stated that the tube should be not less than 35 ft. from the face under the conditions of the test. This distance from the face would vary with each change in volume and velocity of the air coming out of the tubing.

In reading the report in Table 1, it shows the fan developing about 1000 cu. ft. in certain tests and about 2000 in other tests, but the velocity at the end of the tubing when the tests were made is not definitely given. In fact, the velocity at the end of the tubing should be one of the factors affecting the rate of diffusion, and it would have made the report more easily interpreted had the velocity at the end of the tubing been given for each set of conditions. In connection with the rate of diffusion, it may be possible on another occasion to liberate a colored gas, smoke or fumes so that the path of the recirculating gases can be better observed. One of the points that impressed me is that given in test No. 4 with the blower fan located at B-2, 18 ft. out by the line of the rib of No. 1 entry and with a blower fan delivering 1010 cu. ft. of air per min.—the same amount of air as that passing in the main air current. There was a circulation of 37 per cent. and with a normal flow of gas equivalent to 1 per cent. due to recirculation the air passing through the fan was increased to 1.06 per cent. Now suppose that the fire boss, in making his rounds, had found that a large body of gas had accumulated at the face of one of these entries, one could readily realize that a very dangerous condition would exist unless the volume of air passing the blower fan was sufficient to prevent any recirculation. Gas will travel in a body long distances before it will diffuse and if the fire boss is overzealous in speeding up the clearing of the face from gas, the gas may be moved in a body to another blower or some other nonpermissible electrical equipment. Similarly, should the power be off for a few minutes an accumulation may take place during the shift. If one-half of the air is

recirculated the amount of gas is doubled, while if 75 per cent. of the air is recirculated the percentage is four times normal flow of gas.

I wish that the authors in preparing this article for publication would include a curve showing this feature, as a curve of this nature would be a very impressive picture for the use of fire bosses, mine foremen, and others having blower fans in their mines.

While the conditions of the tests are given, I believe more emphasis should be given to conditions to which the tests may not apply, such as pitching beds, thin and thick beds, width of room, and obstructions by cars, timber and gob.

G. S. RICE, Washington, D. C.—This problem has been coming up more and more acutely with the increased mechanization in mines, and we interrupted some of our other testing at experimental mines to make these tests. You will notice they are called "some tests," and as Mr. Greenwald indicated, we feel it is a problem that is by no means finished. There are many other conditions of tests which give different results.

It was agreed that the authors would not discuss fully the dangers of the auxiliary fans. One danger was mentioned by Mr. Smith last year;³ that is, the serious danger arising in the use of auxiliary fans in gassy mines. Generally these auxiliary fans are not operated except on the shift when the men are at work in the rooms or entry and where there is a gas feeder at the face there may be a dangerous accumulation of gas in the interval of 8 or 16 hr. Thus the fan may have to move a large body of gas through working places when the working shift comes on duty.

The authors showed that conditions would be very different if there was a car at the face which would interrupt and prevent the stream of air discharged by the tubing, which without obstruction would strike the face for distances up to 40 or 50 ft. beyond the end of the tubing. Also, in a commercial mine we would have the situation of baffling the current by miners at work. Such obstructions to the air movement must be considered in using the data obtained.

Let us assume that the blower is giving off by undercutting the machine striking it. It may not have the chance of diffusion in the current. In that way you may get a body of gas to face approaching the explosive point.

There should be some discussion by those who have used these fans, to help us in the future.

H. P. GREENWALD.—There is one more point which it is well to keep in mind. Note Fig. 2, and assume that a car is present which will obstruct the end of the tube and prevent the thorough mixing of the gas and air. There will be a higher percentage of gas near the face, but the percentage recirculation at the fan will not be changed. The fan itself is in the air current and the eddy currents caused by it and its general operation determine how the air will divide at the opening into the dead end. Recirculation is not determined at the face, but at the fan. Questions concerning the face are safety questions, not recirculation, and I purposely refrained from discussing safety questions.

R. D. HALL, New York, N. Y.—This paper meets an important need of the coal industry, yet we still desire to learn just how far a recirculating current will go back along a roadway under certain definite conditions. Perhaps the industry will take too much encouragement from the results the authors have set forth. The engineer may say, "The Bureau of Mines showed that there was no gas when the fan was set back 8 ft. from the side airway provided only 40 per cent. of the air was taken, consequently I will put the fan back only 8 ft." It would seem better to put the fan back 16 or even 25 ft. so as to have a factor of safety against the return of gas to the fan.

³H. I. Smith: *Op. cit.*

One should point out that in most of these experiments an unusually large percentage of the main air current is taken by the fan. In many mines there will be 75 men in a split, each requiring perhaps 200 cu. ft. per min. Thus there would be 15,000 cu. ft. of air passing. If, of this, the fan was taking 1000 cu. ft. per min. it would be using only one-fifteenth of the air in the main current, or about 7 per cent. instead of 40 or 50 per cent. —a small proportion indeed.

There are mines where there is not much air in the split, but I would not think that to be true of Pennsylvania, for instance. Of course, it is well to show the auxiliary fan in its worst light but when we do so we should remember that we are so doing.

G. S. RICE.—Necessarily, when you are making a scientific study you have to consider extremes and not averages. I hope that there would always be as much air current in coal mines as Mr. Hall says there is, traversing the last crosscuts, but it has been my experience and observation that such is the average case. We know many instances where the auxiliary fans have been used in rooms turned off entries ahead of the normal air current.

R. D. HALL.—If the place to be ventilated by the auxiliary fan drew air from the last part of the split that might be true, but in a gassy mine there would be gas there and it would not be safe to run the fan in such an atmosphere. We should not have to consider such a case, for it is ruled out for entirely different reasons. In a gassy mine the fan should run in intake air which is free of gas.

G. S. RICE.—We accepted the facts as determined by the tests described in the paper. The results are a little more favorable than any of us anticipated, but the paper does not take into account the many difficult conditions found in practice.

E. McAULIFFE, Omaha, Nebr.—We use booster fans, but have not attempted to put them in gassy mines, and I doubt the wisdom of doing so. The Bureau's studies represent, in my opinion, a distinct contribution to mine ventilation, inasmuch as it has been said that the work of mechanization is traveling, in some districts at least, with considerable rapidity. We did not know just how best to install our booster fans when we started with them. Perhaps we installed some of them wrongly. We changed, however, upon the suggestions of the Bureau, and I am inclined to believe that where any material amount of methane is given off, there is a distinct danger in using the blower fans. This is particularly true if the fans are used in series; that is to say, three or four fans in one entry. We do not get 15,000 cu. ft. of air in every entry in many mines.

There are two or three other practical points I would like to develop; one is the careless way in which fans are installed and the tubing maintained. For example, when we planned to put ours in the entry, back some distance from the working place, no provisions were made in the beginning to carry the tube around the 90° angle, and as a result it was collapsed and choked, giving us the maximum interference and friction.

We also had difficulty in getting our men to extend the tube to the face as the work advanced. I refer now to men working with the shaker conveyors and the "Duck-bill." They objected to the stream of cold air pouring on their backs, and so they would intentionally fail to extend the tube, so much so that we had to resort to disciplinary measure. Those are practical features that deserve consideration, and they are important.

J. A. GARCIA, Chicago, Ill.—I do not believe in the use of booster fans for ordinary practice, but have found in the last year or so that there are certain conditions which practically compel their use in gaseous works. Machines have been developed in this country in the last two years that will drive 30 ft. of entry per shift. Operated on a

two-shift basis this means 60 ft. a day and that is actually being done today. Under the law we would be compelled to put in a crosstee every day which is a ridiculous and an impracticable thing to do. A line brattice is sort of a makeshift and you cannot get the right amount of air to the face with a long-line brattice, so we are compelled under these conditions to use the booster fan, especially in gaseous workings. It would not be possible to design any code that would absolutely prohibit their use unless we could also provide an alternative.

J. J. RUTLEDGE, Baltimore, Md.—Do you use the blower in any other way than as a blower? In some parts of the country the blower is used as an exhaust, of course using the noncollapsible tube.

H. P. GREENWALD.—We did not make any experiments with suction fans. The tube was 12-in. dia. An anemometer at the end of the tube would indicate a velocity of about 1400 ft. per min. If you deliver 2000 cu. ft. per min. the velocity is doubled.

J. KEELY, Kayford, W. Va.—It seems to me that the question as to the amount of air in the entry should be decided one way or the other. Why would not there be 15,000 cu. ft. in the entry if there is 30,000 or 40,000 cu. ft. in the split? It would seem reasonable to expect 30,000 or 40,000 cu. ft. in the entry.

A. C. CALLEN, Urbana, Ill.—How was the air measured at the restrictions, and do you think the restrictions had any influence on the character of the recirculation? What I mean particularly, is this: when the air comes down in the return it is shooting through the orifice at a fairly high velocity. Is it impinging against the opposite rib and perhaps entering the fan?

H. P. GREENWALD.—During air measurement the area was 16 sq. ft. During tests the area was larger and no velocity effects could be detected. Observation of the recirculating current showed that it traveled near the roof against the main current which was concentrated in the lower part of the entry. We traced this recirculating current 100 ft. back of the fan but I do not believe that its outer end could be determined exactly. Any obstruction, such as the body of a man taking samples, would alter the direction of the recirculating current and so change the point where it joined the main air current.

(Written discussion).—Mr. Hall desires to know just how far a recirculating current will go back along a roadway. We found it difficult, if not impossible, to make such a determination, which is fundamentally a determination of the extent of eddy currents in the main ventilating stream. Any obstruction introduced in the stream changes the course of these eddy currents and consequently their extent, and this applies to the body of the man who is attempting to investigate them. However, when the auxiliary fan was taking the entire air current it was evident that the recirculating current along the roof was passing at least 100 ft. to the rear of the fan.

Replying in more detail to the question by J. J. Rutledge: We did not make any experiments with the fan used in any way other than as a blower. The use of exhaust fans requires rigid tubing and we were not provided with this. We did make one experiment not reported in the paper, merely to satisfy our personal curiosity. In this experiment the fan was placed near the face and the mouth of the tubing was in the main air current. With this arrangement an explosive gas-air mixture was obtained at the face in a short time under conditions that would not have produced such a mixture with the fan in its normal position. This arrangement was evidently dangerous and not to be used.

The curve mentioned by Mr. Smith was prepared after the paper had been printed. It is reproduced as Fig. 6. This curve illustrates the increase in gas content caused

by recirculation in a place ventilated. It shows that an explosive mixture can be obtained under conditions which depend on the quantity of recirculation and the percentage of gas present if recirculation is absent.

Mr. Smith dwells more on application of the results than on the experiments by which the results were obtained. It is difficult, if not impossible, to list the conditions

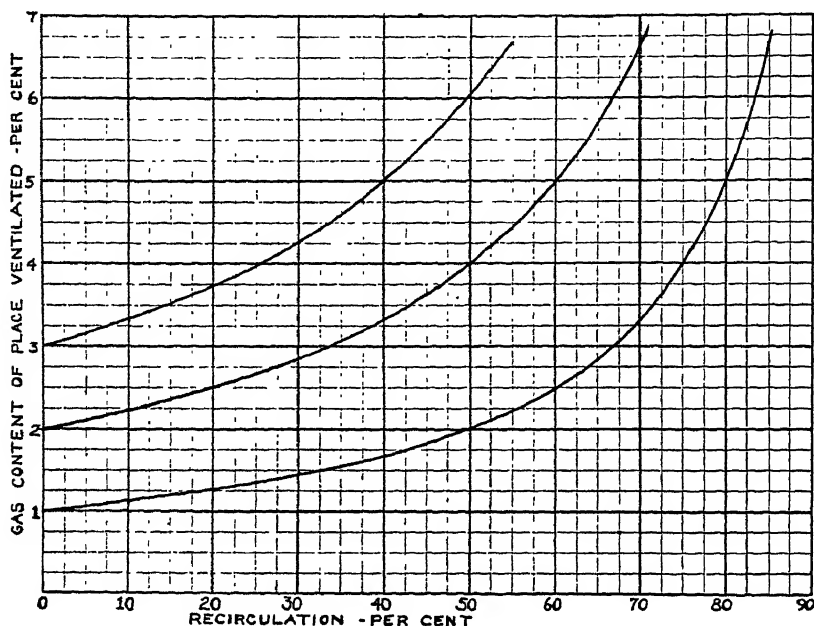


FIG. 6.—RELATION OF GAS CONTENT TO RECIRCULATION.

to which the results of these tests are applicable. Large-scale experimental work can never cover all the variations met in industry and should aim to investigate only those common to a majority of cases. Allowances for variations must be made by the person applying the results and it is at this point that the utmost powers of good judgment are required.

The Classification of Coal

BY CLARENCE A. SEYLER,* SWANSEA, WALES

(New York Meeting, February, 1928)

THE object of all classification is to group together things which are alike, and separate those which are unlike. This object is essentially a practical one, enabling us to apply past experience to new conditions. Many costly large-scale experiments lose half their importance for want of proper description of the coal used, and the consequent impossibility of predicting that other coals will behave in the same way.

I do not, therefore, make a fundamental distinction between "scientific" and "use" classifications, except in so far as use depends on conditions, such as size or impurities which are not related to the nature of the coal substance itself.

The characters chosen to define the classes of coal must be such as are accompanied by as many other properties as possible. It is not to be expected that any system of classification will enable us to predict every property of coal, since all specimens have an individuality of their own. But specimens can be grouped by a number of resemblances into species. Further, species which resemble each other can be grouped into genera.

ELEMENTARY COMPOSITION AS A BASIS OF CLASSIFICATION

There can be little doubt that the ultimate or elementary composition of "pure coal" is the best basis of classification. It was soon after the perfection of the method of elementary analysis of organic substances by Liebig that Regnault, the distinguished chemist and physicist, laid down the principle that coals of the same kind vary only between narrow limits of elementary composition. This is still the widest generalization that can be made about coal. Upon it the metallurgist Gruner based his classification.

In some quarters the value of the ultimate composition was denied, Stein going so far as to say that it teaches us nothing of importance about coal. One still hears statements that it is like crushing a work of art in a mortar and analyzing the powder, or like counting the number of times a given letter occurs in a sentence. Such comparisons are misleading, and the criticism might be applied to organic chemistry as a whole. Nor is the existence of isomerism, or of entirely different substances which

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approach coal in composition, but have not the geological or physical characters of coal, germane to the question. In coal we have a series of natural bodies derived from vegetable matter which has undergone a progressive change known as change of "rank," accompanied by accumulation of carbon and loss of oxygen and hydrogen, and these variations of composition are correlated with the accompanying change of properties.

In 1900,¹ by plotting reliable analyses on a rectangular carbon-hydrogen graph, I verified Regnault's thesis for European coals and based a classification on it. Such a classification depends on the establishment of a correlation between the composition of coal and its properties. The chief of these are (1) the calorific value, (2) the length and nature of the flame, and (3) the caking properties. Of these, the calorific value has only become reliable since the adoption of the bomb calorimeter. The length and character of the flame has long been given a roughly quantitative expression by the determination of the "volatile matter" (exclusive of moisture), but has only recently been standardized. The measurement of the "caking properties" has not even yet been satisfactorily accomplished.

GRAPHIC METHODS OF INVESTIGATION

The graphic investigation of the composition and properties of coal showed in 1900 that we require at least two independent variables to define the position of a coal in the natural series. This accounts for the failure of the earlier attempts at classification based on the volatile matter only. It also showed that any single ratio, such as that of hydrogen to carbon, is unsatisfactory as a basis of classification. Coals differing as cannel and lignite may have the same volatile matter, and cannel and anthracite may have the same calorific value. On the other hand, a classification like that of Parr² on both volatile matter and calorific power will assign a coal to a definite place.

The reason for this, as shown very clearly by Ralston³ in 1915, does not appear to be well known in Europe, nor, indeed, as well as it deserves, in America. Ralston, using what is practically the same graphic method employed by me in 1900, has established for American coals a very fair quantitative correlation between the elementary composition and two of the chief properties of coal on which we have to rely for classification; viz., the calorific value and volatile matter.

The relation between elementary composition and calorific value follows from Dulong's formula. When expressed graphically on the

¹ C. A. Seyler: *The Chemical Classification of Coal*. *Proc. South Wales Inst. Engrs.*, reprinted in *Fuel in Science and Practice* (1924) 3, 15.

² S. W. Parr: *The Classification of Coal*. *Jnl. Ind. & Eng. Chem.* (1922) 14, 919.

³ O. C. Ralston: *Graphic Studies of Ultimate Analysis of Coal*. U. S. Bur. Mines *Tech. Paper* 93 (1915).

carbon-hydrogen diagram the lines of equal calorific value, "isocals," form parallel straight lines, inclined to the axes. Dulong's formula is only approximate, but Ralston showed that if the experimental values are plotted as "isocals" they still form inclined parallel straight lines, though rather differently spaced.

The further correlation of volatile matter with elementary composition, and particularly with the hydrogen, was made by me in 1900, and served as one of the bases of my classification. In 1907, I suggested that for British coals not too widely different in oxygen the relation took the form $H = a + b \cdot \log V$, when H is the percentage of hydrogen and V that of volatile matter in the pure coal, and a and b are constants.

The "isovols," or lines of equal volatile matter in this formula, would be straight lines parallel to each other and to the carbon axis. The same conclusions may be drawn from the results of Fieldner and Selvig⁴ who in 1918 established curves connecting the hydrogen and volatile matter of American coals. They found, however, that the curves differed for coals of different calorific value, in such a way that the isovols would be straight lines successively nearer the carbon axis as the calorific value decreased or oxygen increased.

The discontinuity thus implied is improbable, and Ralston had already drawn the isovols as continuous lines sloping in the opposite direction to the isocals. Ralston says that the lines are not parallel or even straight, but I believe that parallel straight lines represent the relation within the rather wide limits of variation for which it holds good, and they are so presented in Fig. 1. The isovols slope in the opposite direction to that of the isocals and therefore intersect the latter at a definite carbon and hydrogen, as Ralston noted. This is a fact of great practical importance since, as will be seen from Fig. 1, it is possible to infer the carbon and hydrogen from a knowledge of the calorific value and volatile matter, or conversely one can deduce the calorific value and volatile matter when the ultimate composition is known. The isocals or isovols, considered separately, cut through widely different classes of coal, but their intersection on the diagram is a unique point characteristic of one kind of coal alone. Hence, if they can be drawn accurately, any system of classification based, like my own, on elementary composition can at once be translated into terms of a classification based, like Parr's, on calorific power and volatile matter.

Herein lies the possibility of a unified classification which shall be acceptable to those who use either classification. It is unfortunately impossible to compare the limits of my classes of coal directly with those of Parr by means of Ralston's results, because we all use different bases

⁴ A. C. Fieldner and W. A. Selvig: Use of the Hydrogen-volatile-matter Ratio in Obtaining the Net Heating Value of American Coals. U. S. Bur. Mines *Tech. Paper* 197 (1918), 5.

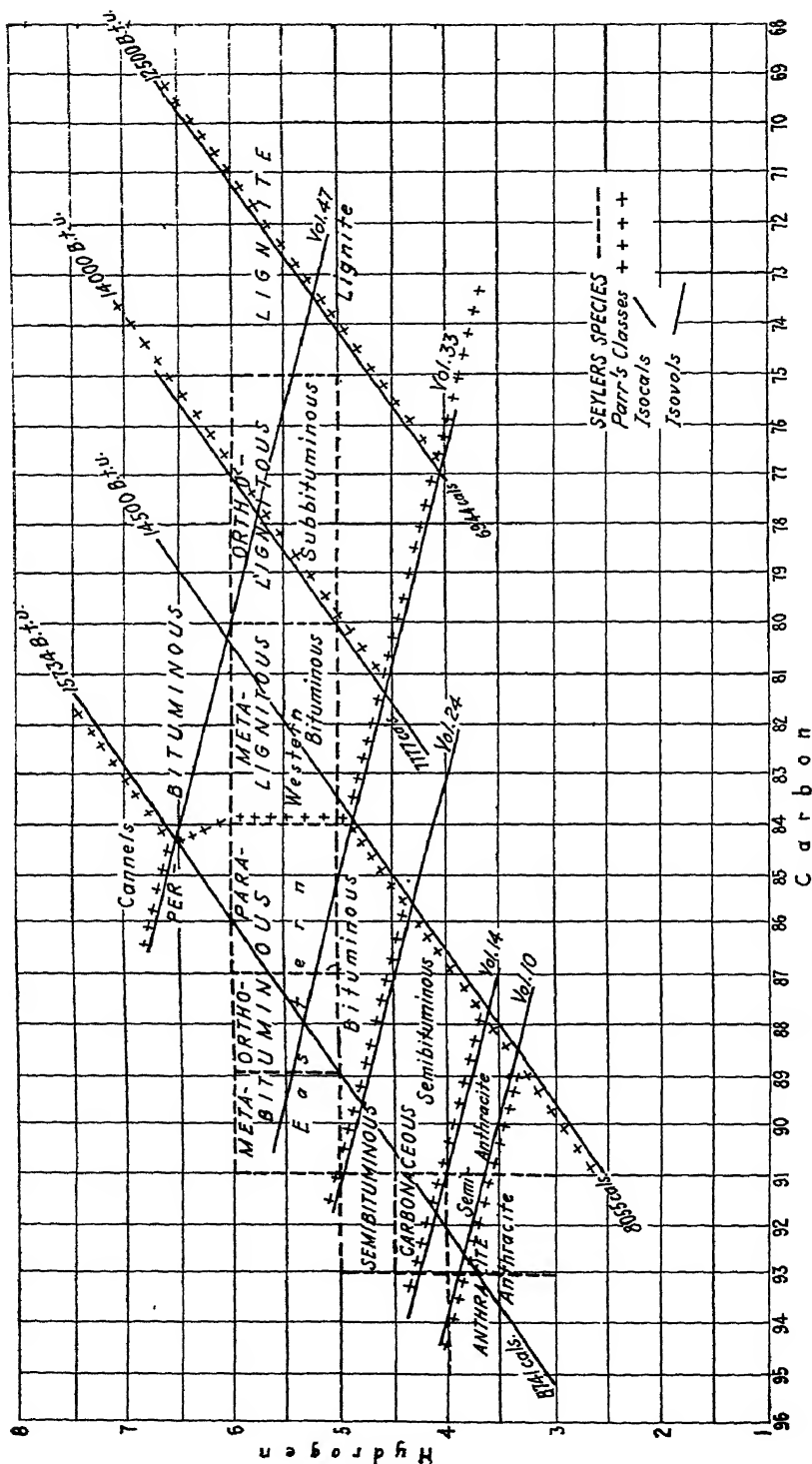


FIG. 1.—CHART OF SEYLER'S AND PARR'S CLASSIFICATIONS OF COAL.

on which to calculate "pure coal." I think, however, that it will be found that the limits between Parr's classes will not differ much from those which I drew in 1900.

NOMENCLATURE

The question of nomenclature requires careful consideration, but I do not think there are insuperable difficulties in the way of unification. We agree on the terms anthracite and semibituminous coals, though the limits can be drawn only after thorough investigation. Between the two is a class which is known in Britain as "dry steam coals" and apparently in America as semianthracitic coals. I call them "carbonaceous" coals because they are the class which, next to anthracite, contains the most carbon. Parr's "eastern bituminous" class appears to correspond to what I call bituminous coals, divided into three groups, metabituminous, orthobituminous and parabituminous, representing approximately low, medium and high volatile coals, suitable for coking. His "western bituminous" coals correspond to what I call "metalignituous" coals. Obviously, the terms eastern and western bituminous are unsuitable for universal use, and if my terms are not acceptable others will have to be found. The chief difficulty in nomenclature occurs with what I call ortholignituous coals, so called because they are the true coals which approach most nearly in properties to lignite.

It is unfortunate that in America these coals, which used to be called "black lignites," have recently been called subbituminous coals. I had previously used this term for coals which were below the bituminous ones in respect to *hydrogen* (subhydrous coals), while those above them in hydrogen (such as the cannels) I called perbituminous. In American usage the term subbituminous suggests coals below the bituminous in respect to *carbon*. This leaves the coals below and above the typical "coal band" of Ralston in respect to hydrogen, which have very distinctive properties, without correlative names such as I gave them, "subbituminous" and "perbituminous." Some way out of this confusion should be sought. One way would be to call all my subbituminous coals semibituminous, and distinguish them as semimetabitanous, semiorthobituminous and semiparabituminous. If the American usage of the term subbituminous is regarded as too well established as the equivalent of my ortholignituous class, then it would be necessary to devise some new term instead of metalignituous to describe those coals which form a transition between the bituminous and subbituminous classes. Such a term as "transbituminous" may be suggested.

When we reach the lignites there is again agreement between our nomenclatures, and I do not think my dividing line of 75 per cent. of carbon between true coals and lignites will be far from the mark. It should be noted however that the Germans use the term "lignit" in a

different way from the rest of the world. They apply it only to the lowest rank of "Braunkohle." What they call Braunkohle we call lignite, though here again there is confusion, since there are certain kinds of Braunkohle (usually distinguished as Glanz-kohle, Pech-kohle, or Glanz-Pechkohle) which we would consider to be true coals (Steinkohlen) although not of Paleozoic age.

It is clear, I think, that there is in the main a general agreement between British and American ideas as to the groups into which coals should be classified. It may be said that, since we have a series of natural objects which vary continuously, it is impossible to draw any but arbitrary lines of division. This is true of many scientific classifications, but there is no reason why, because there is a gradual transition from one kind to another, the chief classes should not be distinguished and named. .

DELIMITATION OF CLASSES OF COAL

The drawing of the limits, however, will have to be to some extent a matter of convention, and will require close study of other properties of coal in addition to the calorific value and volatile matter, especially the caking properties. I do not regard any of the proposed methods of measuring these properties as satisfactory, neither the Campredon nor its modifications, nor the Meurice tests. They are all empirical and not based on any real understanding of the properties which condition the coking of coal. Of these properties that of softening or transient fusion (as Audibert calls it) is the most important, though many other considerations such as rate of heating, and size of pieces, enter into the question of coking, as Rose has pointed out. Strange to say, there is little exact knowledge of the behavior of coal when heated. Experiments on the lines of Dr. White and Miss Stadinchenko with the micro-furnace seem to me in the right direction. There is also the work of Durand, Charpy and Audibert, which I am applying to British coals. It is too early yet to give any details but it appears already that there is a close relation between the composition of coals and their thermal behavior, which will be capable of expression by continuous lines on the carbon-hydrogen diagram in such a way as to help in drawing the limits between the classes. Such at least is my hope. Until more is known of the caking properties of coal it may be premature to draw the dividing lines in a classification in any but a provisional way.

MICROSCOPIC CHARACTERS OF COAL

I must also express the conviction that no system of classification can neglect the microscopic character of the coals. It may be necessary to restrict the chemical comparison of coals to those of similar microscopic character. We do not at present know sufficient to dogmatize on the

subject, but there is certainly a correlation between the microstructure of coal and its chemical and thermal properties which I am now investigating by the aid of the metallographic method. Hence the thorny question of the nomenclature of the physical and microscopic characters of coal requires full and impartial discussion, and I hope it will be possible to reconcile the British and American ideas on this subject.

CORRECTION OF ANALYTICAL RESULTS TO A "PURE COAL" BASIS

There is a previous question relating to all chemical classification which calls for immediate discussion. That is the method of expressing the results on a "pure coal" basis. There can be no comparison of chemical classifications as long as some chemists express their results on the dry ash-free coal, and others eliminate also the sulfur, or the nitrogen, or both. In Ralston's graphs, for instance, both sulfur (uncorrected) and nitrogen are eliminated. My own classification is based on the elimination of ash and sulfur, but not nitrogen. Parr's classification is based on his "unit coal" in which mineral matter (corrected ash) and sulfur calculated to pyrites, have been eliminated, but not nitrogen. Since nitrogen is an essential part of the coal I can see no theoretical advantage in its elimination, and some practical disadvantages, since the available data do not always include its determination. On the other hand, it is essential that pyritic sulfur should be eliminated. Since, however, the relative proportions of organic and pyritic sulfur are rarely available, I suggest that Parr's unit coal basis should be adopted as a convention. This should not prevent those who have complete data from correcting their results in the fullest possible way, but where such data are not available a convention like Parr's, if not wholly satisfactory, is much better than no correction. Failure to eliminate the sulfur is responsible for repeated conclusions that there is no relation between composition and properties.

Possibly some modification of Parr's formula may be required, such as a deduction of an average estimated figure for organic sulfur and the treatment of the rest as pyrites. In any case the heat value of the sulfur as pyrites must be deducted from the calorific values. Moreover, a deduction for loss of pyritic sulfur should be made from the volatile matter, and should be experimentally investigated. The hydrate water due to mineral matter must also be deducted from the volatile matter. These corrections, though often small, may reach important magnitudes, especially in anthracitic coals where the volatile matter is low and small differences important. Further, the hydrogen equivalent to the water existing as hydrated mineral matter must be deducted from the organic hydrogen, a point which Parr seems to have overlooked. The value to be taken for the hydrate water in the mineral matter requires further investigation, but for immediate use Parr's formula (modified if necessary)

is suggested. In the practical application of any classification we cannot limit ourselves to coals of the highest purity, and therefore provision for corrections must be made.

IMMEDIATE PROBLEMS OF CLASSIFICATION

To conclude this imperfect survey the first task should be agreement on the mode of expressing the results as pure coal. Following this the results, recalculated on this agreed basis, should be plotted on the carbon-hydrogen chart, and the isocals and isovols redrawn both for American and other coals, where thoroughly reliable data (and these alone) are available. When this has been done the thermal behavior, caking properties, etc., should be plotted as far as is possible. It will then be found, I think, that coals will fall into groups, definable by the composition, which will correspond fairly with commercial and technical usage. Grouping by fundamental properties is, however, far preferable to any attempt to classify by use, since use depends on these properties, and use classifications are liable to overlap, as is the case, for instance, with coke and gas coals.

Finally, the modifications necessitated by microscopic investigation of individual bands in the coal should be discussed and the question of nomenclature then considered. I am convinced that a standard classification must be based on really definite chemical data such as the ultimate composition and calorific value. The determination of the "volatile matter" is a conventional assay depending on complex and ill-defined changes. It can be used in conjunction with the calorific value, as shown above, to place coals approximately in position on a standard classification. Such approximation is only rough and insufficient for scientific purposes, and it is surely time that all industrial processes should be conducted on scientific lines.

DISCUSSION

A. C. FIELDNER, Washington, D. C.—Mr. Seyler has presented the arguments for the scientific classification of coal on the basis of ultimate analysis in a convincing manner. Even though we should not agree on the system of nomenclature, we must agree upon the scientific facts presented which are in entire agreement with those of Ralston of the Bureau of Mines. Both Seyler and Ralston published their first papers on the classification of coal by ultimate analysis before the industry was ready for such classification. Seyler's first paper was published in 1900. Ralston's technical paper was published in 1915. In this paper he predicted that the trilinear diagram showing the relative proportions of carbon, hydrogen and oxygen in coals would be used by those interested in the coking properties, the gas-making properties, combustion and other methods for utilizing coal. Mr. Rose has given evidence that this diagram is very useful in predicting coking properties of coal. I am, therefore, inclined to agree with Mr. Seyler that there is no fundamental distinction between scientific and use classification except in so far as use depends upon conditions such as size or impurities which are not related to the nature of the coal substance itself.

The fact that Seyler and Ralston are in agreement with respect to drawing "isovol" and "isocal" lines on the ternary diagram simplifies the work of the Technical Committee on Scientific Classification of Coal. The plotting of analyses of American coals with respect to ultimate analyses and with respect to the "isovol" and "isocal" lines will permit direct comparison between Parr's method of classification of coal and the diagram of Ralston's method. One of the immediate problems before the Committee on Classification, as pointed out by Seyler, is the question of correcting analytical results to a pure coal basis. I am not sure, in my own mind, whether correction of the ash for change in weight due to sulfur, carbonates, water hydration, etc., is necessary for classification. Tideswell and Wheeler also discuss the various corrections and are in substantial agreement with Seyler that these corrections should be made although they recognize the fact that the corrections are themselves only approximations. They suggest another method of avoiding the influence of errors due to inorganic matter. This method consists of subjecting the coal to a float-and-sink test eliminating all but the intrinsic ash. Usually the intrinsic ash is small in amount, not exceeding 3 to 6 per cent. With this amount, the analytical results do not require special corrections. Our Committee will necessarily thoroughly investigate this subject of correcting analytical results. The suggestions made by Seyler, and by Tideswell and Wheeler² are extremely valuable.

G. H. ASHLEY, Harrisburg, Pa.—A question that comes up in regard to pure coal is, shall we attempt to say what is pure coal until we know more about the condition of water in the coal? In a paper³ on the condition of water in serpentine, P. G. Nutting points out that the water exists in three distinct forms—what may be called the adsorbed water, the monohydric condition and the dihydric condition, the water under something like 368° being adsorbed water, the water between 368° and 620° being in the form of dihydric water, and the water between that and complete dehydration, about 900°, being in the form of monohydric water.

My studies of the general problem have convinced me that we simply do not know the actual condition of water in coal any more than we know the actual condition of water in cellulose, and until we do, it seems to me we are talking with one very important element left out—if we are talking about pure coal and take no account of the condition of the water which may be in one or two or three different conditions in the coal.

R. THIESSEN, Pittsburgh, Pa.—There is one question to which I would like to call attention and that is in regard to what the Germans at present call lignite, or "lignit." In the last year or so I have been in quite close contact with the German coal investigators. There is a group of young men who are working quite extensively on the various phases of their coal problems and on the classification of coal as well.

What they call "lignit" corresponds exactly to the ingredients or constituents that I call anthraxylon. Lignite to them only means woody inclusions in any coal. The ordinary lower rank coal they call "Braunkohle," which they divide into various classes according to origin or, in other words, according to their components, and inclusions of woody substances in those coals are called "lignit." They have the earthy brown coals, ordinary brown coals and lignitic brown coals. The earthy brown coals have hardly any apparent woody inclusions. The ordinary brown coals have

² F. V. Tideswell and R. V. Wheeler: See page 200.

³ P. G. Nutting: Some Mechanical Properties of Moist Granular Solids. *Jnl. Wash. Acad. Sci.* (1927) 17, 185.

quite a number of woody inclusions, which, as I said, they call "lignit." They also have coals composed almost entirely of woody matter mostly of coniferous origin.

W. FRANCIS, Washington, D. C.—If we assume, and I think Mr. Seyler has shown pretty well that such an assumption is justified, that the ultimate analysis of English coals will give a fair measure of their rank and will classify them according to their general properties, and if we also assume that American coals may also be classified according to some such system, are we safe in assuming that the same classification would apply to American coals that applies to British coals? I would like to suggest that of the different geologic coal-forming ages, there may be one system for Tertiary coals, one for Cretaceous coals and one for early Carboniferous coals, but it does not follow that these systems of classification can be exactly correlated one with another.

Mr. Fieldner has raised the point of inaccuracy and the corrections that have to be made with ultimate analysis. In view of the many errors that may occur in analysis, and the very small range of analyses of all of our coals it would seem that Seyler's classification contains too many fine distinctions and too many names. He has used tools which are too crude accurately to define the properties of coal. Ultimate analysis can only give approximations and will not specify the exact nature of any coal.

H. J. ROSE, Pittsburgh, Pa.—In our laboratories we have made an extensive graphic study of many hundreds of coals, both American and foreign. We have a chart 15 ft. long showing 600 United States coals, and other charts showing analyses given by Mr. Seyler for Welsh, French and German coals. It has been very gratifying to compare the location of coals of known behavior on these different charts. We have frequently asked foreign visitors to point out two or three coals produced in their country which they consider best for a given purpose. For example they will point out several coals and say, "We consider those to be typical of our best gas coals." Turning to the chart of the American coals, we will find them to be similar in ultimate analysis to coal from Logan County, West Virginia, Wise County, Virginia, eastern Kentucky, or other well-known gas coal fields.

In the same way we can compare coals for other purposes. However, there are a great many ultimate analyses of European coals for which I have as yet found no American duplicate, especially low-hydrogen coals and a few high-hydrogen coals. In some cases, errors in the foreign analyses seem probable. However, coals in the same location on the chart have, in general, shown similar properties. It has been remarkably encouraging.

C. A. SEYLER (written discussion).—I regret that I was unable to be present at the discussion of my paper. It is satisfactory however to find that Mr. Fieldner endorses the general proposition arrived at independently by myself and Ralston, that the ultimate analysis is a sound basis for classification, and that it can be correlated with one based upon volatile matter and calorific power by a careful plotting of the "isovols" and "isocals."

I regard it as the highest testimony to the value of such a classification that Mr. Rose has been able by its means to group together coals of similar coking properties, whether they are of French, British, German or American origin. This is in itself a partial answer to the question raised by Mr. Francis as to whether different classifications will have to be made for coals of different geologic ages.

I admit that the problem of the correction for ash, sulfur and water presents difficulties, but this applies to any chemical classification. I do not think it could be avoided by float-and-sink tests since there is a danger in such methods of segregating the banded constituents of coal which differ in specific gravity and ash.

My view is that with standard methods of analysis and corrections the ultimate analysis is the best basis of chemical classification. I do not hope that it will enable

us to predict *all* the properties of a given coal, but it is the best means of placing a coal in a definite position in that curious transition from unaltered vegetable matter to anthracite which we speak of as change of "rank."

I cannot agree with Mr. Ashley that our want of knowledge of the exact conditions under which water may exist in coal should prevent us from expressing the results of coal on a dry, ash-free basis, just as we do with cellulose and other organic substances. If this difficulty had led us to express the results of elementary analysis upon moist instead of dry organic substances no correlation between composition and properties would ever have been possible.

In conclusion I have urged the need of careful microscopic study of coals in addition to the chemical investigation.

Pure Coal as a Basis for Classification

BY F. V. TIDESWELL* AND R. V. WHEELER,* SHEFFIELD, ENGLAND

(New York Meeting, February, 1928)

THE suggestion, which appears to find increasing favor, that the elementary composition of coals should be used as the basis of their classification, makes it important that our methods of expressing this elementary composition should be refined. It is doubtful whether many published ultimate analyses of coals are accurate within 1 per cent. carbon. Many are probably as much as 3 or 2 per cent. in error in carbon. It is, perhaps, not always realized that the extreme variation in carbon content over the range of bituminous coals is but 16 per cent. An error of 1 per cent. in carbon on analysis corresponds therefore with one-sixteenth of the whole range, an amount which is unjustifiable even for the application to commercial needs of a system of classification based on ultimate analysis. For the purposes of scientific research such limited accuracy is of little value.

The object of this paper is to examine one important source of error in ultimate analysis, that due to the occurrence in coal of inorganic material, and to emphasize the need for the quantitative measurements (of carbon more particularly) on coals to be rendered independent of their contents of such material.

ADVENTITIOUS MINERAL MATTER

When a determination of calorific value, for example, is made upon a coal, it is customary to correct the measured value with respect to the ash left by the coal on incineration, recording the result as being on an "ash-free basis." It is desirable, before discussing the adequacy of the correction thus made, to demonstrate its validity, which turns on whether the ash is present as an adventitious impurity in the coal and forms no part of the coal substance itself.

The inorganic matter associated with coal has originated in several ways¹:

1. The "inherent" inorganic materials, occurring as constituents of the plants from which the coal-forming deposits were derived.

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¹ R. Lessing: *The Mineral Constituents of Banded Bituminous Coal. Jnl. Chem. Soc. (1920) 117, 256.*

R. A. Mott and R. V. Wheeler: *Fuel in Science and Practice (1927) 6, 416.*

2. Material deposited contemporaneously with the dying plants, generally under water and hence intimately admixed with them.

3. Material infiltrated into or deposited in crevices, or on layers in the plant accumulations after their deposition.

The material of Classes 2 and 3 forms the "adventitious" mineral matter of the coal.

The adventitious mineral matter can in no sense be regarded as forming part of the true coal substance. Its occurrence in the coal is incidental to the manner of deposition and the later treatment of the coal deposit. Much of this material, particularly of Class 3, can readily be removed mechanically.

INHERENT MINERAL MATTER

The inherent mineral matter in coal represents the inorganic constituents of the original plants. The composition of such constituents of the plants of the coal age was probably similar to that of the mineral matter of present-day plants, *i. e.*, it was rich in lime, silica and alumina and contained lesser amounts of alkalies and magnesium, with only traces of iron. During the formation of coal, much of the water-soluble minerals was leached out, so that the inherent mineral matter of the final coal is composed mainly of silica and alumina, and is reduced in quantity. In general the inherent mineral matter of most coals would seem to be less than 1 per cent. Vitreous, which are almost free from adventitious mineral matter, frequently contain under 1 per cent. of ash. Some durains, free from infiltrated salts, but contaminated with co-deposited mineral matter, contain between 1 and 2 per cent. of ash, while Mott and Wheeler² have shown that, by fine grinding and flotation, the ash content of coals containing a high percentage of mineral matter intimately mixed with them (so that ordinary methods of washing do not remove it) can be reduced to nearly 1 per cent.

This small amount of inherent mineral matter, though a necessary ingredient of the coal, forms no part, so far as is known, of the molecules of the coal substance. From our present viewpoint it is, like the adventitious mineral matter, but a chance impurity in the coal-forming material. We may therefore group together the whole of the mineral matter in coal as being foreign to the coal substance³ and can conclude that it is justifiable to make a proper correction to analytical data whereby they can be referred to the "pure coal."

² *Loc. cit.*

³ There is the possibility that materials derived from the alkaline earth salts of the ulmin acids may be present in coal. No evidence of such salts in coal has so far been obtained and normally the amount of lime in the coal available for such compounds is small in amount.

COMPOSITION OF PURE COAL

This pure coal is composed of carbon, hydrogen, oxygen, nitrogen and sulfur, all elements forming the organic compounds of the original plants. The inclusion of nitrogen and sulfur in the pure coal should, in our view, be unquestioned. Nitrogen and sulfur form a necessary part of the molecules of the most important coal substances, the ulmins, from which they cannot be removed without complete destruction of the molecules.

To obtain an accurate estimate of the composition (or quantitative property) of the pure coal, it is necessary first to separate the contributions of the pure coal and of the associated compounds (moisture and mineral matter) to the observed analytical values. Then, the contribution of the pure coal must be related to the actual amount of pure coal present. The correction as usually made for ash content (*i. e.*, coal = 100 - ash, per cent.) cannot be accepted as adequate, since it assumes that the mineral matter in the coal and the ash left by it on incineration are identical in amount. Actually, the ash is almost invariably less in amount than the original mineral matter, so that the correction applied is too small. The error so introduced is frequently such as to affect the carbon content by 1 to 3 per cent. Further, the correction for ash usually disregards any carbon or hydrogen contributed by the mineral matter, while the common method of making a separate correction to the coal analysis for the total amount of sulfur present, though convenient, cannot be justified.

"Unit Coal"

These considerations were put forward many years ago. Parr and Wheeler⁴ suggested the term "unit coal" for the pure coal substance and derived formulas for the correction of the analyses to the basis of the pure coal. In making allowance for the mineral matter from which the ash was derived, these authors recognized the effects of shaly material containing water of hydration, of calcium carbonate and of pyrites, and assumed, for ease of calculation, that the whole of the sulfur of the coal was present as pyrites. Strahan and Pollard⁵ discussed in detail the relation between the different types of original mineral matter and the amounts of ash obtained from each by incineration of the coal, but contented themselves with pointing out the liability of error involved in the usual correction for ash on analyses. Krym⁶ has also discussed in detail

⁴ S. W. Parr and W. F. Wheeler: Unit Coal and the Composition of Coal Ash. Univ. Ill. Eng. Exp. Sta. *Bull.* 37 (1909).

⁵ H. Strahan and W. Pollard: The Coals of South Wales. *Mem. Geol. Surv. England and Wales* (1915).

⁶ V. S. Krym: *Izvesty & Teploteahn. Inst.* (1926) 3, (16) 18.

the necessary corrections, but makes no application of his formulas to the several hundred analyses listed in his paper. The small attention paid to this earlier work and the almost universal neglect to apply the suggested corrections make a reexamination of the subject necessary.

TYPES OF MINERAL MATTER

Though the mineral matter associated with coal is complex and varied, it is in the main composed of three types of materials: (1) Shale and clay residues; (2) pyrites and related materials (such as sulfates); (3) carbonates (present as partings).

1. *The clay residues* associated with coal contain a considerable proportion of water of constitution which is lost on heating. The residue loses nothing further on incineration, so that the correction to be applied is purely for this water of constitution. This probably varies in amount in different coals and it is unfortunately not usually possible to determine directly the actual content of water of constitution in the clay residues associated with any particular coal. Except where such a direct determination is practicable, it is necessary to accept a mean value which will enable reasonably good correction to be made.

The water associated with ordinary clays is of the order of 10 to 20 per cent.; that associated with coal clays will probably be rather less. Mott and Wheeler⁷ gave 10 and 8 per cent. for the combined water of the shales associated with two samples of Yorkshire coals, but pointed out that these shales differed considerably from the mineral matter separated by flotation from the coals themselves. The values given by Mott and Wheeler for the combined water of mineral matter separated from coal are perhaps high because of water taken up during the process of separation. After very careful drying, combined water contents of 12.9 and 12.4 per cent. have recently been obtained.

Parr and Wheeler⁸ by comparison of the calorific values and ash contents of fractions separated by "float and sink" tests from a sample of impure coal, deduced a value of 8 per cent. for combined water content of the mineral matter. This value was intended to include other losses on ignition, and gave satisfactory correction of the observed calorific value over a wide range.

We have recently had the opportunity of comparing a large number of analyses of a coal seam of variable ash content but uniform composition. On plotting the ultimate compositions against the ash contents, a straight line graph is obtained, indicating uniformity of composition both of the pure coal and of the ash yielded by associated mineral matter

⁷ R. A. Mott and R. V. Wheeler: *Loc. cit.*

⁸ S. W. Parr and W. F. Wheeler: *Loc. cit.*

on incineration. This ash yield was 90 per cent. of the mineral matter. Direct determinations on the roof shale, after correcting for admixed coal, gave 91 per cent. of ash, and 9 per cent. of combined water. For this coal, therefore, about 9 per cent. of combined water is present in the associated shale. The same value (9 per cent.) can conveniently be taken as a mean for most coals. This value corresponds to an ash content of 91 per cent. Therefore the total shale is given by (shale ash + $\frac{9}{91}$ shale ash) = 1.10 shale ash.

The correction for this is important when a coal of high ash content is under consideration. The carbon content is particularly affected, while the hydrogen content must be further (and oppositely) corrected for the water yielded by the shale. In the following examples (Table 1), taken from actual analyses, the corrections are made thus:

Mineral matter = $(1 + w)$ ash. Therefore pure coal = $100 - (1 + w)$

$$C_0 = \frac{100C}{100 - (1 + w)} \text{ and } H_0 = \frac{100(H - \frac{1}{9}w \cdot \text{ash})}{100 - (1 + w)}$$

where $w = \frac{\text{combined water of shale}}{\text{ash of shale}}$

and C_0 , H_0 represent the carbon and hydrogen contents of the pure coal.

The inadequacy of the ash-free correction is apparent. The increase in carbon obtained on correction to mineral matter-free is proportional to the assumed percentage of combined water in the mineral matter. Pro-

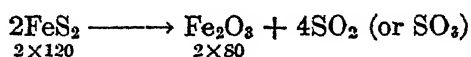
TABLE 1.—*The Effect of Added Coal Shale on Coal Analyses*

Material	As Analyzed			On Ash-free Basis		Mineral Matter-free Basis					
						Combined Water 5 Per Cent. on Ash		Combined Water 10 Per Cent. on Ash		Combined Water 15 Per Cent. on Ash	
	C, Per Cent.	H, Per Cent.	Ash, Per Cent.	C, Per Cent.	H, Per Cent.	C, Per Cent.	H, Per Cent.	C, Per Cent.	H, Per Cent.	C, Per Cent.	H, Per Cent.
Coal A	71.8	5.36	1.5	72.9	5.45	73.0	5.44	73.0	5.44	73.1	5.43
Coal A + mineral matter separated from coal....	65.4	4.98	9.6	72.3	5.51	72.7	5.49	73.1	5.46	73.5	5.44
Difference				-0.6	+0.06	-0.3	+0.05	+0.1	+0.02	+0.4	+0.01
Coal C	83.7	5.50	0.6	84.2	5.54	84.2	5.54	84.3	5.54	84.3	5.55
Coal C + clay	75.8	5.08	9.2	83.5	5.59	83.9	5.57	84.3	5.54	84.8	5.52
Difference				-0.7	+0.05	-0.3	+0.03	nil	nil	+0.5	-0.03
Coal B	88.5	4.40	1.9	90.2	4.49	90.3	4.48	90.4	4.47	90.5	4.47
Coal B + shale from same coal	81.2	4.09	9.5	89.7	4.52	90.2	4.48	90.6	4.45	91.1	4.42
Difference				-0.5	+0.03	-0.1	nil	+0.2	-0.02	+0.6	-0.05
Coal B + mineral matter from another coal	82.4	4.22	7.9	89.5	4.59	89.9	4.56	90.3	4.53	90.7	4.50
Difference				-0.7	+0.10	-0.4	+0.08	-0.1	+0.06	+0.2	+0.03

vided a value of between 5 and 15 per cent. is chosen for this, the maximum error involved is less than that introduced by the simple ash-free correction. If a mean value, say 9 per cent. combined water (*i. e.*, 10 per cent. on the shale ash), is assumed, the maximum error possible is about one-half that of the ash-free correction. These relative amounts are unchanged by the actual amount of ash present, but the absolute magnitudes of the corrections and errors involved are proportional to the ash content. The estimate of 8 per cent. combined water made by Parr and Wheeler is substantially in agreement with our own. The assumption of 10 per cent. combined water on the ash gives, however, more ready calculation. Should more accurate correction of a high-ash coal be desired a determination of the constitutional water of the associated mineral matter would be essential. The necessity for this would arise only in specialized research.

2. *Pyrites and Related Materials.*—The iron in the mineral matter of coal is almost wholly present in combination with sulfur, originally as iron pyrites, approximately FeS_2 . From this may be derived, by oxidation, ferrous and ferric sulfates; and, by interaction, other sulfates. The amount of iron and other sulfates (except in coals which have undergone pronounced atmospheric oxidation) is normally small and for our purpose can be neglected. We will assume that the iron and inorganic sulfur are present, combined as iron pyrites, FeS_2 .

Pyrites on complete combustion yield ash approximately according to the equation:



and thus should yield 66.7 per cent. of ash. Actual determination on a sample of lump pyrites from coal gave 67.1 per cent. of ash. Since, then, the actual amount of pyrites present is 1.5 times the amount of iron oxide yielded on ashing, any considerable amount of pyrites present will materially influence the composition of a sample of coal when calculated to the ash-free basis. Thus, 3 per cent. of pyrites associated with 97 per cent. of pure coal would appear as 2 per cent. of ash. The carbon content of the coal calculated to 98 per cent. of pure coal would be 0.7 to 0.9 per cent. lower than that calculated to the 97 per cent. of pure coal actually present. That is, an error (defect) of about 1 per cent. in carbon would arise for each 3 per cent. of pyrites present. Many coal samples, as analyzed, contain 3 to 6 per cent. and even more of iron pyrites.

Table 2 shows the effect of added pyrites on the composition of coal, and indicates the advisability of correcting to a pyrites-free, and not merely an ash-free, basis. For convenience somewhat high proportions of pyrites were added for these tests. In actual practice, perfect correction cannot readily be made, since the elimination of oxides of sulfur

during ashing may be rendered incomplete by interaction with bases present in the ash. Such retention of sulfur is generally only of importance in the presence of considerable amounts of bases such as lime, and even then the error introduced is seldom as much as one-third that involved by neglecting altogether the elimination of sulfur, in the usual way. Generally, the error introduced by interaction is much smaller.

TABLE 2.—*Influence of Added Pyrites on Coal Analyses*

Sample	As Analyzed			On Ash-free Basis		Mineral Matter Present			Analysis on Mineral Matter-free Basis	
	C, Per Cent.	H, Per Cent.	Ash, Per Cent.	C, Per Cent.	H, Per Cent.	Shale,* Per Cent.	Pyrites,† Per Cent.	Total, Per Cent.	C, Per Cent.	H, Per Cent.
A.....	71.8	5.36	1.5	72.9	5.45	1.65	Nil	1.65	73.0	5.44
A + pyrites.....	64.5	4.82	9.0	70.9	5.30	1.5	11.5	13.0	74.1	5.52
Difference.....				-2.0	-0.15				+1.1	+0.08
C.....	83.7	5.50	0.6	84.2	5.54	0.7	Nil	0.7	84.3	5.54
C + pyrites.....	76.9	5.00	6.1	81.9	5.32	0.6	8.35	8.95	84.5	5.49
Difference.....				-2.3	-0.22				+0.2	-0.05
B.....	88.5	4.40	1.9	90.2	4.49	2.1	Nil	2.1	90.4	4.47
B + pyrites.....	80.6	4.10	7.5	87.1	4.43	1.9	8.65	10.55	90.1	4.56
Difference.....				-3.1	-0.06				-0.3	+0.09

* Shale assumed to contain combined water equivalent to 10 per cent. of shale ash.

† The proportion of pyrites is calculated from the ash content of the sample as analyzed, owing to the difficulty of obtaining a uniform sample of pyrites and coal. Where subsidiary interaction of the ash constituents may raise the total ash beyond that calculated, this method will indicate slightly greater pyritic content than will direct determination and will therefore increase the error due to this interaction. Hence the somewhat overcorrected values of Coal A.

The overcorrection of Coal A (Table 2) is due to this cause, but is emphasized by the special method of calculation used for convenience in this instance; direct determination of pyrites in the sample, or of iron in the ash, would have given a lower value and reduced the amount of overcorrection.

Correction for Pyrites

This brings us to the manner in which the correction for pyrites should be made. (1) Where direct determination of pyrites in the sample is possible, correction may be made directly, thus: Total pyrites = $1.875S_{pyr.}$; pyritic ash = $1.25S_{pyr.}$. Addition to ash for pyritic sulfur correction = $0.625S_{pyr.}$. This method has the further advantage that determination of the inorganic sulfur enables the organic sulfur of the coal to be calculated. (2) A more convenient method, and one sufficiently accurate for the purpose of the correction, is to determine the iron in the ash. This method, though giving somewhat too high values for pyritic content, corrects automatically and adequately for iron sulfates present:

$\text{FeS}_2 = 2.14\text{Fe}$. $\frac{1}{2}\text{Fe}_2\text{O}_3 = 1.43\text{Fe}$. Correction to ash = $+0.71\text{Fe}$ ($\frac{5}{7}\text{Fe}$).

We must emphasize our belief that some such estimation of pyrites present is essential in all coal analyses where appreciable iron is found in the ash. Unfortunately, this determination has in the past been made infrequently, though total sulfur is generally determined. When only total sulfur, comprising both the organic sulfur of the pure coal and the inorganic sulfur, has been determined, it becomes necessary for the purpose of correction to assume the whole of this as pyritic sulfur (as suggested by Parr) or to make an approximate deduction for the organic sulfur. Fortunately, though the pyritic sulfur varies widely, because of its adventitious origin, organic sulfur in coal is much more uniform in amount.⁹ While published data of the organic sulphur content of coals are scanty, we may quote the following collected figures for various coals:

Powell and Parr ¹⁰ (1919), Per Cent.	Yancey and Fraser ¹¹ (1921), Per Cent.	Woolhouse ¹² (1925) Per Cent.
1.25	0.71	1.09
0.43	1.17	1.87
0.47	0.33	0.72
1.33	0.79	0.55
0.55	0.77	1.64
1.66	1.30	0.91
1.24	0.78	0.83
0.21	1.81	0.89
0.63	0.46	0.87
0.61	0.51	0.98
		0.36
		0.23

All the values range around 1 per cent., and therefore (3) in the absence of more complete data, it would be preferable to accept 1 per cent. organic sulfur, as an average value, and assume that the pyritic sulfur amounts to the total sulfur less 1 per cent. $S_{\text{pyr.}} = S_{\text{total}} - 1$. Pyrites = $1.875(S_{\text{total}} - 1)$. Pyritic ash = $1.25(S_{\text{total}} - 1)$ and correction to ash = $0.625(S_{\text{total}} - 1)$. This method should, however, be adopted only when more complete data are unobtainable. Methods (1) and (2) are recommended.

3. *Calcium Carbonate*.—There remains, of the chief inorganic impurities in coal, calcium carbonate (frequently admixed with other carbon-

⁹ H. F. Yancey and T. Fraser: The Distribution of the Forms of Sulphur in the Coal Bed. Univ. Ill. Eng. Exp. Sta. *Bull.* 125 (1921).

¹⁰ A. R. Powell and S. W. Parr: A Study of the Forms in Which Sulphur Occurs in Coal. Univ. Ill. Eng. Exp. Sta. *Bull.* 111 (1919).

¹¹ H. F. Yancey and T. Fraser: *Loc. cit.*

¹² T. G. Woolhouse: The Powell and Parr Method for the Determination of Sulphur Compounds in Coal. *Fuel in Science and Practice* (1925) 4, 454-456.

ates). Calcium carbonate when present is usually obvious, as "partings" or "ankerites." When so present, it can be estimated in the usual way. Normally it may be ignored. Calcium carbonate on ignition loses 44 per cent. of carbon dioxide (the magnesium carbonate frequently associated loses carbon dioxide even more readily and to a greater extent). The elimination (complete or partial) of carbon dioxide occurs during ashing in the muffle or during combustion analysis. When present, therefore, calcium (or other) carbonate introduces two serious (though opposing) errors: (1) The contribution of carbon dioxide, amounting to 0.12 per cent. added carbon for every 1 per cent. carbonate decomposed. (2) A reduction of ash yield of 0.44 per cent., leading to an apparent decrease, on the ash-free basis, of approximately 0.4 per cent. in carbon, for every 1 per cent. of carbonate decomposed.

Assuming complete decomposition, correction could be made thus: If m = carbon dioxide yielded by treatment with acids, correction to ash = $+m$; carbonate ash = $1\frac{1}{11}m$, leaving total ash $-1\frac{1}{11}m$ to be otherwise corrected. Correction to carbon = $-\frac{3}{11}m$.

Tests have shown that decomposition of the carbonate is substantially complete during ashing in a muffle or during combustion analysis on the carbonate alone. During the more careful combustion analysis of a mixture of coal and carbonate (see Table 3) it appeared that elimination of carbon dioxide occurred only partially. Correction to the ash-free basis leads to carbon values too low by 1 per cent. or so for 10 per cent. carbonate present. The error is so small because of the contribution of carbon from the carbonate. On the other hand, correction to carbonate-free, assuming complete elimination of carbon dioxide, gives values both for carbon and hydrogen considerably too high, and therefore cannot be adopted. It would be possible to make further estimation of the carbon dioxide remaining in the ash and to correct accordingly and this procedure

TABLE 3.—*Effect of Added Calcium Carbonate on Coal Analysis*

Sample	As Analyzed			On Ash-free Basis		Mineral Matter Calculated			Analyses on Mineral Matter-free Basis	
	C, Per Cent.	H, Per Cent.	Ash, Per Cent.	C, Per Cent.	H, Per Cent.	Shale,* Per Cent.	CaCO ₃ , Per Cent.	Total, Per Cent.	C, Per Cent.	H Per Cent.
Coal C.....	83.7	5.50	0.6	84.2	5.54	0.7	Nil	0.7	84.3	5.54
Coal C + added carbonate.....	77.6	4.98	6.8	83.3	5.35	1.4	9.1	10.5	85.5	5.55
Difference				-0.9	-0.19				+1.2	+0.01
Coal B.....	88.5	4.40	1.9	90.2	4.49	2.1	Nil	2.1	90.4	4.47
Coal B + added carbonate.....	81.5	4.03	8.2	88.8	4.39	3.0	9.1	12.1	91.5	4.58
Difference				-1.4	-0.10				+1.1	+0.11

* Shale assumed to contain combined water equivalent to 10 per cent. on shale ash.

could be adopted if necessary. Nevertheless, both because of this difficulty of correction and because of the possible interaction of the lime with other ingredients of the ash, it would be safer to insure the absence of such carbonates from the coal sample. This presents little difficulty, because of its manner of occurrence. If mechanical separation, *e. g.*, from a lump sample, is not practicable, float-and-sink separation, particularly from acid solution, will yield a carbonate-free sample. As already stated, most coal samples are practically free from carbonate and need not be so treated.

Summary of Discussion on the Three Types

The inorganic ingredients of coal for which correction should be made on analysis are thus the shale and clay containing combined water and the inorganic sulfur, conveniently regarded as iron pyrites. The actual amount of shale ash cannot be directly determined, in the presence of pyritic ash, but must be determined by difference. The actual correction may be made in several ways, but probably the most convenient is to calculate the actual amounts of pyrites and shale present, thus:

$$\text{Pyrites} = 1.875S_{\text{pyr.}} \text{ or } 2.14\text{Fe or } 1.875 (S - 1)$$

$$\text{Pyritic ash} = 1.25S_{\text{pyr.}} \text{ or } 1.43\text{Fe or } 1.25 (S - 1)$$

$$\text{Shale ash} = A (\text{total ash}) - 1.25S_{\text{pyr.}}$$

$$\text{Shale} = \left(1 + \frac{w}{100}\right) (A - 1.25S_{\text{pyr.}}), \text{ where } w = \text{combined water in shale, per cent. on shale ash.}$$

$$\text{Total inorganic material} = \left(1 + \frac{w}{100}\right) (A - 1.25S_{\text{pyr.}}) + 1.875S_{\text{pyr.}}$$

If the combined water associated with the shale ash is assumed to be 10 per cent., the total inorganic material = $1.1 (A - 1.25S_{\text{pyr.}}) + 1.875S_{\text{pyr.}}$ = $1.1A + 0.5S_{\text{pyr.}}$. The pure coal is then given by $100 - (1.1A + 0.5S_{\text{pyr.}})$.

The carbon values may then be corrected directly to this basis, as may be, if desired, nitrogen and organic sulfur. Hydrogen needs a preliminary correction for water of hydration of shale (assumed to be 10 per cent. on shale ash).

$$\text{Water} = \frac{w}{100} (A - 1.25S_{\text{pyr.}}) = \frac{1}{10} (A - 1.25S_{\text{pyr.}}).$$

$$H_0 = \frac{100(H - \frac{1}{90} \text{ shale ash})}{100 - (\text{shale} + \text{pyrites})} \text{ or } \frac{100 (H - \frac{1}{90} (A - 1.25S_{\text{pyr.}}))}{100 - (1.1A + 0.5S_{\text{pyr.}})}$$

$$\left. \begin{array}{l} C_0 \\ N_0 \\ S_0 (\text{organic}) \end{array} \right\} = \frac{100C \text{ (or } N \text{ or } S_{\text{org.}})}{100 - (\text{shale} + \text{pyrites})} \text{ or } \frac{100C \text{ (or } N \text{ or } S_{\text{org.}})}{100 - (1.1A + 0.5S_{\text{pyr.}})}$$

$$O_0 = 100 - (C_0 + H_0 + N_0 + S_0).$$

In practice, analyses are now generally made on undried samples. The further correction for moisture content should be added to the formulas.

The errors introduced by the presence of inorganic matter, and the necessary calculations to the pure coal, are not confined to the ultimate composition but apply to all other quantitative determinations on coal, *e. g.*, to volatile matter content and calorific value.¹³ The corrections to be applied in reducing to the pure coal basis are similar, with the appropriate deduction for any actual contribution of the inorganic material to the measured value.

ACCURACY OF CORRECTION FOR INORGANIC MATERIAL

Though some space has been given to discussion of the means and manner of correction, where necessary, for the presence of inorganic material, it is not considered that such correction can be made with more than approximate accuracy. Some of the various sources of error in the finally corrected values have been pointed out earlier. More accurate correction, though not impossible, would demand laborious investigation. The correction as indicated, to the pure coal basis, is a practicable one and one which we are finding it necessary to adopt in our work to get reasonably accurate analysis of coal. The errors still involved are proportional to the ash content of the sample. In our view, if the composition of the pure coal is in question, it is essential to reduce the ash content to as low a value as is practicable before analysis, either by careful selection of the sample or by mechanical separation, washing, float-and-sink separation, or even by some method of selective flotation with oil, after fine grinding. Such methods get rid, not only of the carbonate, but of much of the shaly and of the coarser pyritic ingredients. When analyses of coal are required for purposes of classification on the basis of ultimate composition, this preliminary reduction of the ash to a reasonably low value (not exceeding, if possible, 4 to 5 per cent.) is essential to obtain the necessary high accuracy in the analysis after correction to the "pure coal" basis. We anticipate that, ultimately, all coal samples for analysis, whether required for research or for commercial purposes, will receive this careful preliminary treatment.

SUMMARY

Inorganic materials must be regarded as impurities in coal and their presence introduces errors into the composition as found by analyses which are not adequately corrected for by the usual calculation to an ash-free basis. The magnitude of the errors involved is greater than is perhaps generally realized. By calculation to the basis of the "pure

¹³ S. W. Parr and W. F. Wheeler: *Loc. cit.*

coal," that is to say, the coal free from the associated inorganic matter, which is greater in amount than the derived ash, the errors can be reduced considerably. This applies particularly to the shaly and pyritic constituents, the two main inorganic impurities in coal. Calcium carbonate cannot so readily be corrected for and it is advisable (and practicable) to free the sample from carbonate.

The corrections indicated in this paper are not absolute and, though their use minimizes the errors involved in the analysis of impure coal (and is therefore always advisable), it is desirable, when exact knowledge of the composition of a coal is required, to reduce the magnitude of the corrections to be applied by working on as pure a sample of the coal as can be obtained.

DISCUSSION

H. J. ROSE, Pittsburgh, Pa.—Most workers feel that for scientific classification it is generally necessary to calculate coals to the pure coal basis. For practical purposes we want to consider coal as it is, but in many cases we want to consider the combustible material in coal alone.

R. THIESSEN, Pittsburgh, Pa.—A point I want to bring out concerns the statement that the mineral matter can be eliminated up to about 1 per cent. by washing or leaching. Our experience is that it cannot be brought down to less than 3 or 4 per cent. ash in any coal.

A. C. FIELDNER, Pittsburgh, Pa.—In most cases that is true, although some coals are found in which the ash can be reduced to 1 per cent.

W. H. FULWEILER, Philadelphia, Pa.—How about the coals that contain only $1\frac{1}{2}$ or 2 per cent. of ash?

R. THIESSEN.—That is extraordinary coal. I am beginning to believe, and I think it has been shown in various ways, that some of the mineral matter has actually been combined with the coal substance. That is a matter that is very important and should be further investigated.

H. J. ROSE.—I spoke with Dr. Wheeler, when he was last in this country, on the subject of de-ashing coal for the purpose of analysis, and he said they were grinding the coal exceedingly fine in a colloid mill, finer than for the Trent process, for example, and that by doing so they were able to get figures such as were mentioned. I do not know what results we can obtain with American coals when treated that way.

G. ST. J. PERBOTT, Pittsburgh, Pa.—In an investigation of the Trent process, we ground various coals wet in a ball mill until the average particle size was about 10 microns. On subsequent treatment by the Trent process we found that very little additional ash separation was obtained by grinding finer than 200-mesh (average particle size 35 microns).

H. J. ROSE.—We encountered a difficulty years ago in determining the ultimate analysis of coke. There is not much oxygen in coke but for some work we were on we wanted to determine the amount accurately.

We found that the oxygen figure as ordinarily determined did not mean much; in fact, even the best analysts sometimes reported a negative percentage for oxygen. We decided to take the ash out of the coke by acid extraction. We ground the coke to 200

mesh or finer and extracted it with hydrochloric and hydrofluoric acids and reduced the ash to below 1 per cent. We did this with coke from American coals.

It may be that this method is open to some objection. We did not follow it up very far. Neither of the acids used is an oxidizing acid of the type of nitric acid. This is one possible way of going at it if you are willing to take the time—it is rather tedious—and risk altering the coal with the acid treatment.

G. H. ASHLEY, Harrisburg, Pa.—Apparently, oxidation is likely to result in grinding the coal so fine. We compared the analyses of coal of various sizes from the same anthracite mine and found that the very small sizes of anthracite were above 20 per cent. volatile. The only explanation we could make was that there was oxidation in the very small sizes.

W. H. FULWEILER.—In our experience in attempting to reduce the ash in coals, we have had the best results with the float-and-sink method. We have not had such success with the oil flotation method. We do not believe that it is possible to make a dogmatic statement as to the minimum ash in coal as it depends very much on the type of coal.

Recently, on one sample, we reduced the ash from 7.2 to 1.2 per cent., grinding only to 100 mesh. It seems likely that with finer grinding a lower figure would have been obtained.

A. C. FIELDNER.—There is one point in this paper that may lead to confusion in further work on coal classification. The paper says that the inorganic matter originated from:

"1. The 'inherent' inorganic materials, occurring as constituents of the plants from which the coal-forming deposits were derived.

"2. Material deposited contemporaneously with the dying plants, generally under water and hence intimately admixed with them.

"3. Material infiltrated into or deposited in crevices, or on layers in the plant accumulations after their deposition.

"The material of classes 2 and 3 forms the 'adventitious' mineral matter of the coal."

I believe our practice in this country has been to class only the materials under 3 as adventitious and to classify as inherent those under 1 and 2; that is, the ash coming from the plant plus the ash resulting from material deposited in intimate association with the plant. Obviously the contemporaneously deposited silt cannot be separated by the usual methods of separating out adventitious ash material such as the float-and-sink separation with heavy solutions or the Trent process.

We should bear in mind this difference in nomenclature from that to which we have been accustomed. I should prefer to call classes 1 and 2 inherent and class 3 adventitious.

H. J. ROSE.—That is a point well worth bringing out. There is one question that I would like to ask. In the case of coals which even after thorough cleaning still contain 5 per cent. or more inherent ash, can you see that ash in the coal under the microscope at high magnification?

R. THIESSEN.—If you take a thin section of coal, thin enough to be transparent and examine it under the microscope, no ash is visible. Should you burn this section carefully, you will find that the ash stands out a very delicate colloidal aggregation, in exactly the same form as it was in the coal. It is in a colloidal condition except for a few of the larger grains. If it is touched the whole structure sinks together. If you put it into water, the whole mass floats out in a colloidal suspension, except the few larger grains, and forms a milky liquid. If a drop of this is put under the ultramicroscope, the minute particles are then visible.

H. G. TURNER, Bethlehem, Pa. (written discussion).—Pure coal as a basis for classification is no doubt highly desirable for the purposes of scientific research. To obtain pure coal, however, by making corrections for impurities will, I believe, lead to errors fully as serious as those arising from our present method of analysis. The correction for 8 to 10 per cent. water of constitution for shales would probably be too high in some cases. We find, for instance, that Clarke¹⁴ gives a composite analysis of 78 shales as follows:

	PER CENT.
Water at 110° C.....	1.34
Water above 110° C.....	3.68
Total.....	5.02

Kemp gives these four analyses:

L. V. Pirsson, of Yale University, gives:

TOTAL WATER PER CENT.	TOTAL WATER PER CENT.
3.84	8.1
6.26	9.7
6.51	2.7
0.20	8.5
4.17 Average	7.25 Average

These analyses show a range of 0.20 to 9.7 per cent. with the average for combined water falling far below the 10 per cent. suggested for the purpose of correction.

If we assume that the argillaceous matter is clay rather than shale, we come closer to the mark, as is shown by the following clay analyses:¹⁵

WATER FROM CLAYS DRIED AT 100° C.
PER CENT.

4.63	1.95	
10.46	2.02	
11.72	4.11	6.23 Average
10.79	3.21	

Here we find a range of 1.95 to 11.72 per cent. with an average of 6.23 per cent., which is again lower than the assumed 10 per cent.

In anthracite coal where the argillaceous material approaches a slate, the water percentages would be less than 4 per cent.

Assumed relations of sulfur to iron are also unsafe as is shown by the ash analysis of the Red Ash coal from Panther Creek, Pennsylvania, where 4.71 per cent. sesquioxide of iron is found in an ash containing only 0.285 per cent. sulfur.

Again, our knowledge concerning the inherent ash of coal is very scant. Whereas the ash content of modern trees such as oak, beech and pine, falls far below 1 per cent., that of modern representatives of coal-forming plants ranges from 3.25 per cent. in *Lycopodium dendroideum* to 11.82 per cent. in *Equisetum hyemale*. The mineral matter contained in the coal-forming plants was obviously reduced in cases of low-ash coal, while in high-ash coal we do not know how much of the ash is inherent. If we define inherent ash as that portion of coal ash which cannot be removed without chemical alteration of the organic coal substance, we have a far more definite concept.

I concur with the writers' conclusion that coal as pure as possible should be secured for use as a basis for a classification of this kind.

¹⁴ F. W. Clarke: Data of Geochemistry. U. S. Geol. Surv. Bull. 695, 4th ed. (1920)

¹⁵ F. W. Clarke: *Op. cit.*

F. V. TIDESWELL and R. V. WHEELER (written discussion).—We do not think there should be much difficulty in reducing the mineral matter in most coals to a little over 1 per cent. by suitable treatment. It is possible that a small proportion of this mineral matter is combined with the organic matter of the coal, as Dr. Thiessen suggests. It seems to us logical to term "inherent" only that mineral matter that was derived from the inorganic constituents of the original plants. Though Mr. Fieldner states that this term is often taken to include the silt deposited contemporaneously with the plants and equally difficult to separate from the coal, we think "residual" a better term to apply to all such inorganic material, whether "inherent" or "adventitious," as cannot be removed by physical means.

As Professor Turner states, exact correction for the water of constitution of coal shale is impossible because its amount varies so much. From our experience with British coal shales we consider the best mean value to lie between 8 and 9 per cent., a value differing but slightly from the 9.1 per cent. (calculated on the shale) assumed in our paper, which gives a convenient 10 per cent. correction on the shale ash. The error introduced by this assumption would in general be small, certainly smaller than that incurred by complete neglect of the fact that the shale contains combined water. The use of the iron content as a measure of the amount of pyritic sulfur present has been found to be sufficiently exact for most British coals, while it also gives a rough but necessary correction for any iron carbonate or hydroxide.

The methods suggested have now been applied to a large number of British coals and found to give satisfactory corrections. We have therefore adopted the system as standard in our work, though we prefer that the samples of coal analyzed should be as free of mineral matter as possible.

Valuation of Coal Properties

By JOHN B. DILWORTH,* PHILADELPHIA, PA.

(New York Meeting, February, 1925)

THIS paper treats primarily of the valuation of developed coal properties by the method of capitalizing their estimated average future earnings. However, reference is also made to valuations of undeveloped coal lands and to various methods of appraisal other than by future earnings.

FUNDAMENTAL VALUATION PRINCIPLES

Before discussing in detail coal property valuations, a few statements bearing on the general subject will be made to present certain of the writer's views on the question as a whole.

1. A certain property at a given time has but one true value. This is not universally admitted. The appraiser rather frequently finds that a coal owner has in mind three valuations for his holdings; one, the figure at which he feels they should be appraised for taxation purposes; another, the much higher one that should be used if a loan is to be negotiated on the property; and a third, intermediate between the other two, the price for which he would be willing to dispose of his holdings were they for sale.

The last is usually nearest to the true value, and because this is so the statement is generally accepted that the fair value of such a property is the price arrived at by negotiation between a willing seller and a willing buyer who are otherwise disinterested.

2. Notwithstanding the foregoing, it is only very generally and approximately true that the proper measure of the value of a given developed¹ coal acreage is the unit or per-acre price for which one or more bodies of similar coal in the same district have been sold recently.

To give a few reasons why such a measure of value can not be strictly applied, it may be noted:

a. That sales are often forced by circumstances at times when the coal business is depressed and buyers can be found only for properties that are offered at figures far below their true value.

b. That conversely, during periods of unusual prosperity, abnormally high prices are paid. Such conditions obtained several years ago when coal brokers, with very profitable contracts but no sure source of supply, were bidding against inexperienced or shortsighted operators for proper-

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¹ By "developed" is meant coal tributary to an existing mine.

ties in the belief that profits of from one to three dollars per ton on bituminous coal could be obtained indefinitely. During this period absurdly high prices were paid for coal land, which were in no sense a measure of its true worth, as has since been demonstrated.

c. That sometimes the amount of money asked and paid for a property is far more than its normal or intrinsic value, because of considerations other than general utility involved in the transaction. The strategic position of the tract, or its peculiar desirability to a particular party for a special purpose, may have influenced the price very materially.

d. That the outstanding reason why the sale price of one coal area is only a general measure of the value of another, even with the same seams and in the same district, is the fact that rarely are two bodies of coal identical in all respects, or even sufficiently similar to have the same unit value. The truth of this statement will be shown by examples further on in this paper. It is sufficient to say here that, given two developed areas of the same coal bed, identical in thickness, character and position, their unit or per-acre value will vary materially with their accessibility and with their life or the speed with which they will be mined.

3. The true measure of the value of an industrial enterprise, such as a coal-mining operation, conducted solely for profit and without regard to sentiment or personal enjoyment, is the amount and rate of earnings it will afford with intelligent and experienced management.

This fundamental economic truth has been well expressed recently by a prominent financial writer as follows.²

One of the most valuable single concepts which the student may obtain from the study of economic principles is that value is derived from income, not income from value. As Professor Fisher says in "The Elementary Principles of Economics," "The value of capital *must* be computed from the value of its expected future income. Not until we know how much income an item of capital will bring us can we set any valuation on that capital at all. * * * Not original cost, reproduction cost, book value or appraised valuation, but *earning power* is the criterion."

As an evidence of the truth of this conception in its application to developed coal property valuation, it will be observed that most of the items in the following schedule of salient features affecting the appraisal of such a property relate either to the cost of producing coal or to the amount of money that can be obtained for it; the difference between which is the profit to the operator, or the earnings upon the capital representing the worth of the property.

Major Features Determining Coal-land Value

- a. Geographic location of area controlled—its relation to transportation, sources of supplies, labor, markets, etc.
- b. Accessibility of area to mining development.
- c. Assurance of coal reserves—thoroughness of prospecting.

² Walker Van Ripper: Practical Hints for Investors. *Barron's* (Sept. 6, 1926) 5.

- d. Prospective output of mine and life of coal reserves.
- e. Character of coal seams—thickness, partings, regularity, position.
- f. Quality of the coal—rank, purity, heat units, ash fusibility.
- g. Uses and market reputation of the coal.
- h. Mining, preparation and marketing facilities of the operator.
- i. Replacement cost of plant and mine and the sale value of surface owned.
- j. Operating records—past production and earnings.
- k. Labor situation—organized or unorganized; supply.
- l. Terms of leases, if coal be so controlled.
- m. Comparative conditions on lands with competing mines.

SEGREGATION OF INVENTORY ITEMS

Appraisal of a developed coal property by the earnings method gives merely the total value of the entire producing unit and theoretically takes but little cognizance of its component parts. In practice, however, it is customary to segregate certain items of a similar nature (such as plant, surface land, etc.) and appraise them by established methods, giving to the coal the residual value left after deducting the total of these individual appraisals.

Convenient subdivisions of such a property for valuation purposes, and approved methods of appraising them are as follows:

Mine Plant and Equipment, comprising structures, tracks, power lines, drainage and ventilation systems, and their accompanying excavations, fills and grading; also machinery, tools, instruments, supplies and livestock. Such items are usually appraised at the estimated cost of replacing them by new units at current prices of machinery, materials, and labor, less such depreciation for age, condition, obsolescence, and lack of usefulness as judgment and experience dictate.

Mine Development.—This may advantageously be subdivided. Class A covers that of the more permanent type, such as drift portals, shafts, shaft-bottoms, rock slopes or tunnels, main-entry grading and permanent timbering, all of which are appraised in the manner indicated for mine plant and equipment. Class B covers mainly mine entries, air-courses and rooms with their break-throughs, cross-cuts and overcasts, all of which are constantly being abandoned in whole or in part and as continuously being replaced in kind until the mine ceases to advance into new territory.

A proper method of valuing Class B development has never been agreed upon generally by engineers, appraisers, or mine operators; but it is the writer's opinion that the truest measure of the value of such an asset is the difference between the cost of opening out the mine until its workings can furnish the output for which the plant is designed and the normal cost of producing the coal so mined.³ Where there is considerable

³ For a fuller treatment of this subject, see paper by the writer. *Trans.* (1922) 66, 715.

excess development of this class in a mine, additional value should be allowed, based upon its net cost.

Town.—This subdivision comprises dwellings for officials and miners, clubhouses, garages, bathhouses, store and similar structures, with the accompanying fences, sidewalks and streets and the water supply, lighting and sewage systems. These assets are appraised by the usual replacement-cost method mentioned.

General Construction Items.—These are mainly interest on capital outlay during plant installation and mine development, and the cost of engineering, supervision and insurance over that period. The appraisal of this subdivision is based on the amount and rate of capital expenditure during the development period and the interest rate for money.

Surface necessary for plant and town sites is appraised at the current selling price of similar land in the locality, unit figures of which have usually been fairly well established by local sales.

Occasionally a property has far more surface than is needed for mining. In such instances the excess surface should be disregarded in the calculation of total property value by the earnings method and appraised as a separate entity. This is likewise true of special timber tracts or any such asset not related to the mining or preparation of coal.

VALUATION OF COAL RESERVES

In most instances, when a developed property is under consideration, valuation of coal reserves can be made most satisfactorily by estimating the worth of the industrial unit as a whole on an earnings basis and deducting from the total the sum of the appraisals of plant, development, surface, etc., made by methods already indicated.

Considerable judgment and experience are necessary to apply this method properly and the results obtained should be checked by comparison with those given by other methods of procedure. But, recognizing that appraising is not an exact science and that in any method the element of personal judgment must enter largely, it is submitted that the one under consideration involves more reasonable assumptions, permits greater delicacy of adjustment to varying conditions, is more subject to demonstration and contains in itself more automatic checks against error than any other that may readily be employed.

Justifiable Interest Rates

Before proceeding with examples of valuation of coal ownership and lessees' equities, a decision should be reached as to proper interest returns on capital invested in coal-mining enterprises under various conditions of control.

'Take first the case where the entire property is owned by the operator. What may be considered a reasonable average rate of return on his invested capital over the life of the mine?

Bituminous coal operation has fewer risks than most mining enterprises when undertaken by experienced men, guided by competent financial and engineering talent. Most failures in it have been due to unwise selection of properties, lack of experience, and insufficient capital, all of which are preventable. Nevertheless, it has some uninsurable natural and industrial hazards, and in almost every field it is subject to severe competition in the marketing of its product. It follows, therefore, that the rate of return on capital invested in a coal-mining plant and its tributary acreage should be very appreciably higher than that afforded by good bonds or well seasoned stocks, and somewhat above what is considered equitable for capital invested in many manufacturing enterprises not of a "wasting" or self-consuming nature, and where there are possibilities of varying product to suit changing demands of the market.

Experience has shown that in the older coal-mining districts of the United States, where the industry is on a fairly stable basis, an average return of 12 per cent. per annum on capital invested in the coal and plant of an active property may be considered reasonable. And this rate used in valuation calculations gives results according quite well with those obtained by other methods, where these can be applied with reasonable assurance of accuracy.

In the case of a lessee-operator who owns his plant but is mining another's coal under an agreement of lease, the capital is more in jeopardy than if the operator owned the coal and no lessor had first call on earnings for royalty payments. It follows that he should anticipate a higher rate of return on his investment than the fee owner. Where the royalty paid is about one-third of the average earnings from operation, which is normally the case, 15 per cent. per annum is a fair rate of return on the lessee's capital. If royalty payments are relatively low, 13 per cent. interest is sufficient; if high (half or more of probable average earnings), 16 per cent or a higher rate should apply.

Conversely, the coal owner-lessor's capital is more secure than if it were invested in both plant and coal. Usually he has a lien on the lessee's plant to secure payment of royalty and his income is to a degree independent of the fluctuations of operating costs and coal prices. A fair average return on an owner-lessor's invested capital may be placed at 8 per cent. per annum, assuming ordinary conditions to prevail. In unusual cases, where the coal value is greatly in excess of plant value and the royalty amounts to half or more of the average earnings, a return of about 9 per cent. on the coal-owner's capital is more equitable.

It frequently happens that a lessee becomes a lessor and subleases a portion of the coal he controls at a higher royalty than he himself pays

to the owner. Such a sublessor is freed from the direct risks of mining and marketing the coal but he still has to meet his obligations to the owner-lessor, so his equity capitalization should be figured with a somewhat higher interest rate than that of the owner-lessor. Naturally, this rate should vary somewhat with the margin between the owner's royalty rate and his own, but generally an interest rate between 9 and 10 per cent. per annum will be found equitable in such instances.

Finally, the case of the owner whose coal has been leased and then subleased to an operator. He has not only an ultimate lien on the plant, but both lessee and sublessee must fail before his royalty income is disturbed. Moreover, in instances where the owner's royalty rate is well below the normal regional rate for such coal, he may well be able, in case of failure of both lessee and sublessee, to re-lease his property to yield a much greater royalty return than he is now receiving.

His coal holdings therefore approach the nature of a prime investment; though they are usually not quite so, inasmuch as it is rarely beyond the range of possibility that the coal body may develop irregularities or otherwise afford less than the calculated tonnage, and the owner is generally at some expense for supervision of lessee and for local taxes on the coal. All things considered, a fair interest return on capital so invested would seem to be 7 per cent. per annum when the actual royalty rate is about half the normal, and ranging on up to 8 per cent. as the actual rate approaches the normal. The 8 per cent. rate is also justified where the value of the coal leased greatly exceeds that of the mining plant, as will be shown later by example.

APPRAISAL EXAMPLES

Illustrations follow of the application of the estimated earnings method of valuation to concrete cases. These exhibit many of the features of the method more clearly than is possible by abstract discussion.

A type property will first be considered, *i. e.*, one where conditions of coal acreage, bed thickness, yield, mine production, life, earnings, taxes, and the relation of plant cost to rate of production are such as obtain in many fields throughout the country. This type property will be designated as *Class A* and its coal value will be found approximately equal to its plant value.

To show generality of application, two other properties will be considered incidentally: *Class B*, with thin coal, relatively small production and earnings, and a plant value over twice the coal value; and *Class C* with thick coal, large production and earnings and a plant worth about half as much as the coal.

Basic features of these three properties are given in Table 1. Values in the first group are assumed for the individual properties; those in the

last are derived from the assumed conditions by the calculations to follow. Bracketed letters in the table are symbols to be used in a formula given later.

TABLE 1.—*Fundamental Features of Various Classes of Properties*

Items	Class A (Type)	Class B (Cheap)	Class C (Costly)
Assumed			
Coal acreage (horizontal bed).....	800	500	1,250
Coal thickness.....	5 ft. 0 in.	3 ft. 6 in.	7 ft. 0 in.
Coal yield per acre, short tons.....	7,500	5,000	10,000
Total recoverable, short tons.....	6,000,000	2,500,000	12,500,000
Average annual production, short tons.....	240,000	100,000	500,000
Life of mine, years.....	25	25	25
Surface owned, acres.....	100	50	100
(P) { Value of plant, development and supplies.....	\$300,000	\$106,000	\$600,000
Value of surface.....	\$10,000	\$4,000	\$20,000
Amount of working capital.....	\$50,000	\$20,000	\$100,000
(S) Salvage value of working capital and property.....	\$120,000	\$40,000	\$240,000
Average operating profit, ^a per ton.....	40 c.	25 c.	60 c.
(E) Average operating profit, ^a per annum....	\$96,000	\$25,000	\$300,000
(f) Federal tax on net earnings.....	8%	8%	8%
Local taxes on appraised values.....	1%	1%	1%
(r) Fair interest return:			
Owner-operator.....	12%	12%	12%
Owner-lessor.....	8%	8%	9%
Lessee-operator.....	15%	13%	17%
Owner to sublessor.....	7-8%		8%
Sublessor.....	9-10%		11%
(a) 6 per cent. 25-yr. sinking fund payment per dollar.....	\$0.01823	\$0.01823	\$0.01823
Derived			
(C) Value of coal, total.....	\$304,000	\$44,000	\$1,337,500
Value of coal, per acre.....	\$380	\$88	\$1,070
Value of coal, per ton.....	5.1 c.	1.8 c.	10.7 c.
Normal annual royalty, total.....	\$36,305	\$5,507	\$172,458
Normal annual royalty, per ton.....	15.1 c.	5.5 c.	34.5 c.

^a After all charges except depreciation, depletion, federal tax and interest on capital.

Example I

With the conditions assumed for the type property (Class A), what is the coal worth to an operator who owns both it and the plant?

This can be expressed by a general formula, as will be shown later, but solution by trial is perhaps preferable for the sake of clarity therefore it will be given first, using the basic data in Table 1.

Value of plant, surface and working capital.....	\$360,000 (<i>P</i>)
Value of coal @ (by trial) \$380 per acre.....	304,000 (<i>C</i>)
<hr/>	
Total capital investment.....	\$664,000
Salvage value of plant, surface and working capital.....	120,000 (<i>S</i>)
<hr/>	
Capital to be amortized.....	\$544,000

On the basis of these figures, the estimated average annual earnings (*E*) of \$96,000 should pay 12 per cent. (*r*) interest on the total capital invested and an 8 per cent. (*f*) federal income tax on this interest, and afford an annual sinking fund which will return the amortizable capital at the exhaustion of the coal. Tables show that \$0.01823 (*a*) placed annually in such a fund bearing 6 per cent. interest for 25 years will amount to \$1.00.

As a proof that this is the case we have,

12 per cent. interest on \$664,000 capital.....	\$79,680
8 per cent. federal tax on interest earned.....	6,374
6 per cent. sinking fund to retire \$544,000 in 25 years.....	9,917
<hr/>	
Total annual earnings.....	\$95,971

It follows from these calculations that about \$380 an acre, or 5.1 c. per recoverable ton, is a fair price for this coal under prevailing conditions.

The foregoing calculations may be expressed in a general algebraic form and shortened for convenience. Using the bracketed letters in the table as symbols of the various items to which they are appended, the following equation may be written:

$$\begin{aligned}
 E &= r(P + C) + fr(P + C) + a(P + C - S) \\
 C(r + fr + a) &= E + aS - P(r + fr + a) \\
 C &= \frac{E + aS}{r + fr + a} - P
 \end{aligned}$$

Substituting the values assumed in the example,

$$C = \frac{96,000 + 0.01823 \times 120,000}{0.12 + 0.0096 + 0.01823} - 360,000 = \$304,193$$

Coal values in Class B and Class C properties, under the conditions assumed for them in Table 1, may be shown by the calculations or formula used for the typical Class A property to be \$44,000 or \$88 an acre for Class B and \$1,337,500 or \$1070 an acre for Class C.

To indicate some of the ways in which this method of coal valuation is useful to appraisers of developed lands, and to owners and operators, there will be considered first the case of a property identical in all respects with the type one except that it contains but 480 acres of coal with 3,600-,

000 recoverable tons, has a life of 15 years and a salvage value of \$140,000. What is its coal worth?

This may be determined by trial calculations, or by the formula, with sinking fund tables giving $a = 0.04296$, thus:

$$C = \frac{E + aS}{r + fr + a} - P = \frac{96,000 + 0.04296 \times 140,000}{0.12 + 0.0096 + 0.04296} - 360,000 = \$231,180$$

which is equivalent to \$481 an acre, the value of the coal.

Were the tributary coal area only 320 acres, so that the life of the operation should be 10 years, its coal would be worth, by the foregoing method of appraisal and allowing \$150,000 salvage value, about \$508 an acre. Were only 160 coal acres available, giving a 5-year life and, say, \$170,000 salvage value, their unit value would be \$319 an acre.

Considering the opposite condition, if there were 1120 acres of tributary coal, giving a life of 35 years and a salvage value of \$100,000, the coal would be worth \$303 an acre.

A comparison of the foregoing figures shows that under the assumed conditions (and they are approximately those prevailing at many operations) developed coal land has its highest unit value when its life is around 10 years. When the life period is shorter, depreciation on plant becomes excessive; when longer, the interest charges (and local taxes as well) take a heavy toll of earnings and reduce the unit value of the coal. Moreover, an operation designed for a long life, say 30 years, often has an expensive type of plant, with steel, stone and brick structures and costly town improvements, all of which add to the capital investment without proportionately increasing earnings, and they correspondingly lower the unit value of the coal holdings.

Practical issues, such as the desirability of maintaining an efficient organization, holding established markets, providing against irregularities of operation and markets, conserving the potentially long life of a large investment in town and community accessories, being able to borrow capital through bond issues, and avoiding the uncertainty of obtaining suitable new territory at a satisfactory price when desired, usually militate against opening a large mine with a life of but 10 years; so that, all things considered, a 25-year life is about the normal for a modern bituminous coal operation. It is unquestionably true, however, that, given two properties with similar natural conditions, the most profitable—the one affording the highest rate of return on invested capital and in which the coal bears the highest valuation—is the one with simple plant structures, efficient but not necessarily elaborate equipment, and a life not exceeding 15 years.

To illustrate the effect on coal value of an abnormal capital expenditure in plant, assume a property identical with the one here adopted as typical except that its plant has a replacement value of \$400,000 because of expensive shafts, a railroad bridge, or some other costly item essential

to the development of the coal but adding nothing to operating efficiency as compared with the type property's plant. What is its coal worth?

Except for a possible slight increase in salvage value, which may be neglected for the purpose of this illustration, the second property is worth no more than the first one, or merely \$664,000. As the value of plant, surface and working capital is fixed by generally accepted methods of appraisal at \$460,000, the coal takes the residual value of \$204,000, or \$255 an acre, instead of \$380 an acre as in the type case.

Consider now the problem of the owner of the type property worth \$664,000, who feels that by installing a preparation and cleaning plant he can increase the net profit on his coal by 10 c. per ton, making it 50 c. a ton, or \$120,000 annually. How much can he afford to spend on these additions?

By transposing the coal-valuation equation to read

$$P = \frac{E + aS}{r + fr + a} - C$$

and solving with trial values for S , the value of plant, surface and working capital (P) is found equal to approximately \$525,000, which is \$165,000 greater than the original \$360,000 value of this item. Therefore, to increase average net earnings by 10 c. a ton, the owner is justified in investing about \$165,000 additional in plant and working capital.

As a check on this calculation, it may be noted that the extra earnings of \$24,000 a year will afford a sinking fund sufficient to retire the amortizable (not the working) capital in 25 years, pay an 8 per cent. federal tax and yield approximately 12 per cent. interest on the investment.

Example II

The mine operator of the type property owns the plant and surface only and leases the coal from its owner. Local taxes on coal are paid by the owner, those on surface and improvements by the lessee. What is the maximum annual royalty the lessee-operator can pay and earn 15 per cent. on his invested capital?

Capital in plant, surface and working capital.....	\$360,000	
Salvage value of plant, surface and working capital..	120,000	
Lessee's capital to be amortized.....	\$240,000	
Total earnings (\$96,000) plus 1 per cent. local taxes (\$6,000).		\$102,000
15 per cent. interest on \$360,000 capital.....	\$54,000	
8 per cent. federal tax on interest earned.....	4,320	
6 per cent. sinking fund to retire \$240,000 in 25 years.....	4,375	
1 per cent. local taxes on plant value.....	3,000	
Lessee's requisite annual earnings.....		65,695
Maximum royalty charge applicable.....		\$ 36,305

It follows from these calculations that a fair royalty for the coal in this property is about \$36,300 a year, equivalent to a rate of approximately 15.1 c. per ton.

The question now naturally arises: Can the owner of this coal, which is considered worth \$380 an acre, or a total of \$304,000, afford to lease it at 15.1 c. per ton royalty, yielding \$36,300 per year, to an operator who will exhaust it in 25 years, assuming the owner-lessor is entitled to 8 per cent. interest on his capital?

Annual income to coal owner-lessor from royalty	\$36,300	
Annual 6 per cent. sinking fund to retire \$304,000 in 25 years	\$5,542	
Annual supervision charge, land and federal taxes	6,200	11,742
		<hr/>
Annual net income (8.1 per cent. on \$304,000 capital)		\$24,558

It seems, therefore, that about \$380 an acre, or 5.1 c. per recoverable ton, is a fair valuation for this coal, whether it be owned and operated at an average profit of 40 c. per ton or leased at 15.1 c. per ton royalty.

It follows from this line of reasoning that if the appraiser has few data from which to estimate future average earnings of the property under consideration, but knows that 15 c. per ton is the regionally established royalty rate for similar coal which can be developed at a cost of about \$1.50 per ton of annual output for plant and working capital, he has strong evidence that this coal is worth about 5 c. per recoverable ton in the ground. By a calculation similar to the one just made, the appraiser can take any royalty rate and determine the corresponding value of the coal so leased.

This means of checking coal valuations is frequently very helpful, particularly in old mining districts where recent sales have been few but where prevailing royalty rates may be easily ascertained. Usually these rates have been determined by long experience and indicate the combined judgment of both owners and operators as to the true value of their unmined coal.

Calculations similar to the foregoing, but using factors assumed for Class B and Class C properties, show that a fair or normal royalty for the cheap Class B coal is \$5500 per annum or 5.5 c. per ton, while for the valuable Class C coal it is \$172,460 per annum, or 34.5 c. per ton.⁴

It must be noted, however, that in the calculations for these properties, with coal values near the extremes of the valuation range, certain adjustments in interest rates should be made to conform to the rather abnormal conditions. The lessee of the cheap Class B coal pays out only one-fifth of his earnings in royalty, therefore he is almost in the

⁴ It is interesting to observe that in all three classes of properties the value of the coal in cents per ton is very close to one-third of the normal royalty rate; also that from 20 to 25 c. of earnings are necessary to pay proper return on capital invested in plant, surface and working capital; i. e., the ordinary lessee's investment.

position of the owner of the entire property and is entitled to only a little higher rate of interest on his capital than if he were full owner. If the latter is due 12 per cent., a fair interest rate on this lessee's capital would be around 13 per cent. and this figure has been used in the calculations.

As for the lessee of the valuable coal, he is obligated to pay over half of his earnings in the form of royalty to the coal owner and a relatively small shrinkage in earnings below the assumed normal would reduce his profits very materially. To compensate for this extra risk, his rate of return on invested capital has been placed at 17 per cent.

In such an instance, the owner-lessee is entitled to a capital return slightly higher than the normal 8 per cent., so 9 per cent. has been used in the current calculation. The principal reason for this increase in interest rate is the relatively small additional investment in plant (about 50 per cent. of the coal value) which the coal owner would have to make to become an owner-operator and thus be entitled to 12 per cent. interest on his entire investment. To secure this return, Class A coal owner would have to invest 100 per cent. of his coal value in mine plant, and Class B owner would have to invest 300 per cent. Moreover, the very fact that the lessee of Class C coal owner has a relatively less costly mine plant to secure the royalty income makes the hazard of Class C coal owner greater than that of Class A and Class B owners and justifies a higher rate of return on his capital.

Example III

Assume that the type property, Class A, had been leased by its owner years ago, when the coal was less valuable and royalty rates lower than at present, at a royalty of 8 c. per ton to a lessee who subsequently sublet it to the present operator at the current 15.1 c. royalty, the sublessee-operator owning the plant and surface. What is the present value of the owner's equity in the coal if it brings him \$19,200 annually, and capital so securely invested warrants a 7 per cent. interest rate?

A general algebraic formula may be developed, as in the case of Example I, to solve this problem by a single operation, but a sufficiently close result may be secured more readily by trial, thus:

Annual 7 per cent. interest on \$181,000 coal equity value.....	\$12,670
Annual 6 per cent. sinking fund to retire \$181,000 in 25 years...	3,800
Annual supervision charge, land and federal taxes.....	3,200

Annual income from royalty..... \$19,170

If it be true that the foregoing owner's equity in this coal is worth \$181,000 (\$226 an acre) and that the fee ownership of the coal is worth \$304,000, it follows that the first lessee's equity is worth \$123,000 (\$154

an acre) and that his royalty income of 7.1 c. per ton, or \$17,100 annually, should afford him a fair return of between 9 and 10 per cent. on this sum. That this is true is shown by the following calculation:

Annual income from 7.1 c. per ton royalty.....	\$17,100	
Annual 6 per cent. sinking fund to retire \$123,000 in 25 years.....	\$2,242	
Annual supervision charge, land and federal taxes.....	3,000	5,242
		<hr/>
Annual net income (9.6 per cent. on \$123,000 equity value).....		\$11,858

Problems of this character frequently arise in practice. Sometimes the coal land under consideration has been leased and then sublet, as in the example just given; but more frequently there is merely an old lease from the owner to the mine operator in which the royalty rate is distinctly lower than that prevailing at present for similar coal in the region, and the appraiser's problem is to determine the respective values of owner's and lessee's equities in the coal.

To ascertain how the foregoing solution of this class of problems works out under varying conditions, assume that Class A property had been leased at 10 c. per ton royalty and sublet at 15.1 c., giving annual royalty incomes of \$24,000 and \$12,300 to owner and sublessor respectively. Here the owner's royalty is two-thirds of the total, or normal, royalty, instead of about half as in the first instance, so his rate of return on capital should be somewhat greater than before as his status is more nearly that of an owner leasing directly to an operator and entitled to an 8 per cent. return.

Using a 7.5 per cent. interest rate and carrying through the calculation as before, we find that the owner's equity is worth \$216,000 or \$270 an acre. The lessee's is worth the difference between the total value of the coal (\$304,000) and the owner's equity value (*i. e.* \$88,000 or \$110 an acre) and his net income of \$12,300 from royalty will return 9.5 per cent. on this valuation after deduction of sinking fund and supervision charges, and proportional land and income taxes.

Were the old and new royalty rates 12 c. and 15.1 c. respectively, a 7.75 per cent. interest rate for the owner's capital would place his equity valuation at \$254,000, or \$318 an acre, leaving \$50,000, or \$62 an acre, as the value of the sublessor's equity upon which his 3.1 c. net royalty would pay about 10 per cent. interest after proper amortization, supervision and tax deductions.

Turning to Class B and Class C properties, Class B may be excluded from the examples of this kind of calculations, as its royalty rate of 5.5 c. per ton is so small that it would hardly be subdivided between a lessee and sublessee.

Taking Class C property, assume that its royalty of 34.5 c. per ton, or \$172,500, annually, is divided—20 c. per ton, or \$100,000 a year, to the owner and 14.5 c., or \$72,500 annually, to the sublessor. Inasmuch as it

has been shown that this owner, when leasing directly at full royalty to an operator, is entitled to a 9 per cent. interest return on his capital, his equity in the present instance should be capitalized at about 8 per cent. Correspondingly, the sublessor's capital should earn about 11 per cent. Following the form of calculation already employed and determining coal equity values by trial, owner's equity valuation is shown:

Annual 8 per cent. interest on \$862,000 coal equity value.....	\$68,960
Annual 6 per cent. sinking fund to retire \$862,000 in 25 years.....	15,714
Annual supervision charge, land and federal taxes.....	15,300

Annual income from royalty.....	\$99,974
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Sublessor's equity valuation is checked as follows, it being the difference between full value and owner's equity value, or \$475,500:

Annual income from 14.5c. per ton royalty.....	\$72,500
Annual 6 per cent. sinking fund to retire \$475,000 in 25 years.....	\$ 8,668
Annual supervision charge, land and federal taxes.....	11,400
	20,068

Annual net income (11 per cent. on \$475,500 equity value).....	\$52,432
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There are, therefore, good grounds for the statement that, in the instance cited, the owner's equity in this valuable coal land is worth \$690 an acre and the sublessor's is \$380 an acre.

Examples of Special Appraisal Conditions

The foregoing groups of examples illustrate the application of the earnings method to the principal classes of coal-land appraisal usually encountered. Each class has many variations, but practically all of them can be handled by an adaptation of one of the methods outlined, always bearing in mind that, in the last analyses, the true measure of the value of an industrial asset is its earning power.

Certain common variations from standard conditions not considered in the illustrations given are mentioned below.

It frequently occurs that a property in which a mine is operating has sufficient coal to last far beyond the efficient life of the plant at the rate of production that the mine may be expected to maintain. In such a case it is suggested that a sufficient acreage be assigned to the mine to give the plant the life that its condition warrants and appraise it as developed coal. The remaining acreage should be appraised as undeveloped land.

In valuing undeveloped coal land, the earnings method is not directly applicable, for obvious reasons, and the recent-sale-price-of-similar-land measure is usually applied with such skill as the judgment and experience of the appraiser can command. Even here, however, the earnings method is usually helpful, unless the field is generally undeveloped, for by it an estimate can be made of the value of a developed area of somewhat

similar coal land and this will give some basis of valuation for the undeveloped tract.

For instance, if Class A coal of these illustrations is worth \$380 an acre, and there is under consideration a similar body of undeveloped coal in the same region which, it is judged, will not be opened to mining for 20 years, such a body may reasonably be considered worth about \$95 an acre today. This is on the principle that an investment in idle coal land will double itself every decade through accumulating interest and tax charges.

Occasionally a property is examined where the replacement value of the plant is so high, or prospective earnings so small, that appraisal by the earnings method leaves little or no value for the coal. This would have been the case with Class A property, for example, if plant and surface appraisal, with working capital, had amounted to \$660,000, or if indicated average future earnings had been 20 c. per ton. The total value derived from earnings is correct; but the coal, disregarding the plant, probably has *some* sale value, so in practice it is best to appraise the coal at a low but reasonable figure and deduct this amount from the plant valuation. In other words, such a result means that the plant is too costly for coal of such character or else that the land has been prematurely developed.

Sometimes an appraiser finds that a property with good coal has been so loaded down with an elaborate plant and town, the high costs of which can not be proportionately reflected in large earnings, that his calculations give a unit value for the coal which is obviously too low as compared with such values on other properties mining the same seam, having similar lives, and showing equal earnings. In such a case the total value should stand, but plant appraisal should be depreciated for inefficiency until the residual figure represents the true worth of the coal.

This assumed condition affords an opportunity to illustrate the fact that the earnings method of appraisal is not a routine system that can be applied indiscriminately by an inexperienced person, nor is it one the results of which need not be checked by pertinent data from other sources and methods. In the case just mentioned there may be an opportunity to increase production or profits very materially by a comparatively small expenditure of capital in a certain department, such as for additional mine cars, power-plant unit, dwellings, picking table, etc. The increase in total value of the operation occasioned by this additional expenditure may be many times the amount of capital required, because the new equipment may add to the usefulness of that already installed. It is only fair under these conditions that the property appraisal be credited with this prospective increase in value, deducting, of course, the sum necessary to bring about the improvement.

Unusual hazards or uncertainties at a particular property may readily be allowed for in appraisal by the prospective-earnings method through

increasing the interest rate used. For instance, some years ago a serious attempt was inaugurated in the Georges Creek district of Maryland to rework certain of the "Big Vein" areas from which scarcely 50 per cent. of the coal had been mined. There was little question of the amount of coal available and no doubt as to its quality and marketability; but the projected method of mining was necessarily somewhat unique, though it seemed feasible, and the cost of operation was uncertain, though apparently it should not have proved excessive.

In valuing the estimated tonnage reserves, it was manifestly proper to allow for the experimental nature of the enterprise and its inherent uncertainties, and this was done by using the earnings method and taking a 20 per cent. interest rate to represent a fair return on invested capital.

Another illustration of allowing for uncertainties is the valuation of coal available for strip mining in a district where this character of operation has not become established. Even with careful prospecting it is difficult to determine future production costs accurately and the market for coal so mined is apt to be more limited and prices more variable than for coal recovered in the ordinary way. Moreover, estimated profits from strip operations are usually comparatively large, due to the wide margin between probable costs of production and the general market price for coal, a margin which is not always realized.

Under these conditions it is frequently wise to figure such property valuations with an interest rate considerably higher than the 12 per cent. ordinarily used for operations with underground mines.

SUMMARY AND CONCLUSIONS

In considering the valuation of industrial properties in general, and of developed coal properties in particular, it is assumed as fundamental that they have but one true value regardless of the object for which they are appraised.

As regards developed coal lands, this value is only very approximately indicated by the recent sale price of neighboring lands, for the reason that it is rarely possible to find a sale of a coal property sufficiently similar in essential mining conditions to the one under consideration to serve as a measure of its value; and if such a one is found there are frequently incidental conditions peculiar to this particular transaction which invalidate the use of the sale price as a measure of general values.

The true measure of the value of a coal-mining operation is the amount and rate of earnings it will afford with intelligent and experienced management. Practically all of the features of a property which are generally recognized as affecting its value relate either to the cost of producing the coal, or to the price it will bring, and so are intimately associated with earnings.

In practice, it is customary to appraise those portions of a developed coal property which are produced by the industry of man—mine plant, equipment, development, town, etc.—at their cost of replacement at prevailing prices of materials, machinery and labor, with such deductions for age, condition, obsolescence, and lack of usefulness as conditions warrant. Surface is appraised at the estimated cost of similar land in the district. Coal value is the difference between the worth of the entire property, considered as an industrial unit and appraised on the basis of estimated average future earnings, and the sum of the valuations of plant, surface, etc., determined by the cost-of-reproduction method just mentioned, plus the necessary working capital.

Appraisal by the earnings method consists essentially in determining the amount of capital upon which a given annual operating profit will afford an equitable rate of interest after paying the federal income or profits tax and after deduction of a sum sufficient to accumulate a sinking fund which will return at the completion of mining the invested capital less the salvage value of the property.

The principal factors involved in this determination are: amount of available coal; average rate of mining; value of plant, equipment and surface; necessary working capital; salvage value of property at end of life; average operating profits; average federal tax rate; justifiable interest rates on invested capital, and rate of sinking fund accumulation.

Most of these factors usually can be determined or forecast for a particular property without any large element of uncertainty, and of those which are uncertain that of average future operating profits is the only one of major importance. Even here the experienced appraiser is not without valuable guides and checks. Almost always he has for his guidance past records of earnings, either those of the property itself or of other somewhat similar ones in the district, and his familiarity with the property in question and general acquaintance with the coal industry over the period covered by these records enable him to judge whether they represent normal or abnormal conditions. Moreover, current royalty rates in the region are an index of general opinion as to future earnings from coal operation and are a valuable guide to the appraiser estimating anticipated profits.

While it is not contended that appraisals by the earnings method should supplant all others, even for developed properties, and it is readily granted that recent sales prices of coal land and the opinions of owners and such interested parties should be given careful consideration, nevertheless it is submitted that this is the most satisfactory single method usually available and that it is based squarely upon the fundamental principles of industrial valuation.

In addition to the fact that it is as accurate as any appraisal method can be expected to be, involving, as all methods must, the element of

personal judgment and the uncertainties of the future, this one is unusually elastic and affords a ready means of allowing for varying conditions such as life of property, relation between plant cost and tonnage output, lessee control, differences in royalty rates, and extraordinary hazards of operation or marketing.

Another feature of merit in the earnings method of appraisal is that its results are susceptible of demonstration. Often in a report or discussion on valuations the only support given to a statement of coal land value is that it is the appraiser's opinion. While it is recognized that in such matters expert opinion should be deemed more or less self-sustaining, as it is based upon knowledge gained from many sources and ripened by experience and is hardly susceptible of full demonstration, nevertheless it is frequently advantageous to be able to show the method and steps by which the given result was derived, and unwarranted criticism may often be avoided thereby.

This is particularly true in cases where a group of properties belonging to different owners are being appraised for a common purpose, as for a merger or consolidation. In such instances an owner receiving a relatively low unit valuation for his coal (say because of long life) as compared to a neighbor having coal of similar character (but a short life), may feel discriminated against until shown that, as far as earnings and other assumed factors are concerned, he has been placed on an equal footing with his neighbor, and that, assuming these premises, which he will probably agree are sound, the given values follow automatically.

DISCUSSION

E. T. CONNER, Scranton, Pa.—I am willing to agree with Mr. Dilworth that where it is possible to determine a long enough base line of earnings that method constitutes a reasonable way of determining the value of coal properties, but it has been my observation that it is rather difficult to find a base line of sufficient length and dependability to universally use earnings as the foundation upon which to determine the value of a given coal property. All who have been in the business of mining coal can realize what I mean by that. We may adopt an earning level that is first rate for last year or the year before last but is not worth anything this year. So we must discount that method.

I have in mind an instance that excited considerable comment, where an enterprise was set up on the basis of earnings over a four-year period, which some of us might say was a good base line of earnings that indicated it was a reliable proposition as an investment. But it happened that those were four mighty good years—I mean good for the coal man in that there was some real cold weather and people had to burn coal and were willing to pay for it. That same enterprise which was set up on a four-year base line of substantial earnings now is in bankruptcy; the earning line has pretty nearly disappeared. So I agree with Mr. Dilworth that we must take into consideration the history of the whole region, and not only the whole region in which the particular properties are located, but the whole region of competition which that coal must meet in the market. So the measuring stick which seems to me the safest is the one that has the longest life of application, and what is that measuring

stick? It is the bargaining between the owner and the lessee, we will say, where coal has been mined on a lease. That bargaining has arrived at a value per ton of coal produced, the royalty value, which seems to me to be the safest measuring stick to apply.

In the State of Pennsylvania, the law requires that valuation shall be placed upon sales of willing buyers from willing sellers. Then the question is, what constitutes sales? If it is possible to find actual bona fide sales of average coal property, that is the measuring stick which must be applied by the assessor and the engineer. If there have not been, as often happens, sales of properties of any considerable magnitude that might be so used, it seems to me that the next best method of determining values is the one that I have mentioned; namely, the bargaining between willing seller and willing buyer over a long period of years which has crystallized in royalty values.

Having adopted such a measuring stick, the base is established, and to that must be applied the life expected or the number of years over which the coal in the property may be extracted, and then comes in the point made by the author of the paper. What is a lessee's equity?

There it is quite necessary, as Mr. Dilworth says, that the appraiser or examiner shall have had pretty broad experience, pretty wide knowledge of what may reasonably be expected by a lessee after having met his obligations under the royalty rate of his lease, and that judgment is really, in my opinion, the vital feature in determining what a coal property is worth. We have that situation in the anthracite region today, where we are unable to sell more than about 50 per cent. of what can be produced, and the same situation is facing many in bituminous mining. So a hard and fast rule of determining values based upon earnings will have to be taken with large discount even where there is an earning line that extends over several years.

C. T. STARR, Pottsville, Pa.—I would like to have Mr. Dilworth talk a little more on the question of the lessee's equity. I wonder if he could tell us a little more about how he would handle the proposition where a lessee is leasing property, paying royalty for the coal and incidentally paying royalty for the use of the equipment? What would be the equity of the man operating such a property?

J. B. DILWORTH.—I do not know whether I can answer that very clearly or not. In most instances, the man who leases the plant also leases the coal. Is that the case in the instance you have in mind?

C. T. STARR.—No, the man leases coal from one party and leases the equipment from another. What would be the equity of the man operating the property under such conditions?

J. B. DILWORTH.—That shows the infinite varieties of this problem. In the first place, presumably the plant would be leased at a fair rental; that is, the rental would be some equitable proportion of its value. As the rental received by the owner of the plant would be commensurate with the value of that plant, therefore the lessee's equity would have no appraisable value. The lessee's equity is of value, in our opinion, only when he is in a somewhat unique position and has a particularly favorable lease. If coal is normally leasing at 10 c. per ton, and Mr. Jones leases a body of coal at 10 c., his equity has no appraisable value. Similarly, if he leases coal and plant for 25 c. and a proportion of that goes to the owner of the plant as a fair return, I should say again that there is no equity value. If, however, by some reason of changed circumstances, a fair rental for the coal and plant is 30 c. per ton, and he was given the privilege of using that plant and mining the coal for 15 c. per ton, of course there would be a valuable equity and it would be appraised by taking the difference between the two rentals. If he has a margin of 15 c., theoretically he would be able to

turn around and lease to some one else for 30 c., and there would accrue to him 15 c. per ton for having the privilege of this lease. Therefore that 15 c. per ton multiplied by the annual tonnage and capitalized by the life of the lease would give the value of the lessee's equity.

S. A. TAYLOR, Pittsburgh, Pa.—Probably 15 or 20 years ago this very question was asked me by the judge in a case of damages, and I replied by saying, "I have given it a great deal of thought but have never come to an exact conclusion. My judgment is that the value of the leasehold of the property is worth from one-fifth to one-tenth of the value of the profit he could make on it." (This was quietly talking to the judge when he told the jury not to listen.) He said, "What does that mean?" I said, "In this case, I think the man could make 20 c. per ton. I am sure if you would allow him 2 c. per ton for the value of this lease it would be equitable, and I would not object to giving him 4 c." He said, "What would it amount to at 2 c. per ton?" I figured it out and he charged the jury to bring in a verdict for that amount.

I have repeatedly tried to work it out, and I believe that one-fifth of the profit that can be made is the value of the lease to the lessee. The royalty paid by the lessee is not the value of the lease to him. That is the value of the lease to the lessor. When the lessee puts his money, his skill and knowledge into developing the property, that lease has an increased value, and I believe one-fifth of the profit that can be made from that lease would be a fair allowance.

A short time ago I had a case on which I used one-fifth. Fortunately for the parties, I had taken the profit made in the whole district and on the particular mine for five years. I found that it was practically 30 c. per ton, so I gave the value of 6 c. per ton to that leasehold. It happened that the lease was taken at a price 5 c. below the regular royalty rates in that field. I did not know that, but it was confirmed by another engineer. Taken over the period of the life of the mine, our figures were almost identical; one was \$200,000 and the other \$210,000 as the valuation of the lease.

It is a matter of judgment, of course, but I am satisfied that in any case the man ought to have at least 10 per cent. of the profits that can be made over a period of years as a value of the leasehold to him after he puts his money in improvements and his brains and experience in the operation of the mine.

J. A. GARCIA, Chicago, Ill.—After a number of years' experience in appraisal work, I believe that the appraisal value or replacement or reduplication cost of a mine has very little to do with the true worth. The author has expressed my ideas very definitely in saying: "The true measure of the value of an industrial enterprise, such as a coal-mining operation, conducted solely for profit and without regard to sentiment or personal enjoyment, is the amount and rate of earnings it will afford with intelligent and experienced management." That amply covers the whole case in my judgment.

I have appraisals in mind, where the duplication costs went to several million dollars but the properties never have made and never will make a dollar. There is practically no relation in most instances between the physical appraisal value or the replacement cost of a mine and its actual worth as a business proposition.

J. KEELY.—Does Mr. Dilworth consider gross earnings?

J. B. DILWORTH.—What do you mean by gross earnings?

J. KEELY.—There might be a good many unusual expenses and deductions in one connection that might be taken up in a merger or something of the kind and done away with; then you would resort to gross earnings, would you not?

J. B. DILWORTH.—You would, if by that you mean deductions for Federal taxes and interest and so forth; if you mean by gross earnings the consideration of those

elements—in other words, the difference between the average selling price and the average mining cost.

J. KEELY.—That explains it fully.

J. B. DILWORTH.—Those are left out because of those three or four factors that enter into these calculations.

J. KEELY.—Other things might affect it also.

H. LOUIS, Newcastle-on-Tyne, Eng. (written discussion).—Mr. Dilworth has selected for discussion the simplest type of mineral valuation—that of a going colliery—thus avoiding what I look upon as the most difficult aspects of the question; namely those of deferment. I am in general agreement with his method of treating the subject and hold with him that the only correct method of valuing such a property is to calculate the present value of the probable annual profit over the estimated life of the property. I emphatically agree with him that there is only one true value of a given property at a given time, but, as I have pointed out in my book on mineral valuation, this true value can only form the basis of negotiation between the seller and purchaser of such a property, the actual price given depending on whether the former or the latter needs the property most urgently, or upon which of the two is in the most need of money. The price of a property, therefore, is related to its value but not absolutely fixed thereby. Where I disagree with Mr. Dilworth is in the positive nature of his statements; thus, in the examples given, he assumes a total recoverable tonnage, an average operating profit per ton, etc.; strictly speaking these figures represent not the actual tonnage or profit but the calculated most probable tonnage or profit—that is to say, a figure which, on the data before the valuer, is likely to be nearer the true figures than any others that can be named. As I have pointed out in my book, such figures, being the most probable figures, are necessarily affected by a probable error, and in any complete calculation the Laws of Probability must be considered, in order to determine probable maxima and minima and thus to enable the valuer to see within what limits deviation from his calculated value is permissible. Personally, I doubt whether the Federal tax on earnings should be applied to the assumed annual profit, because, if I understand the position correctly, such a tax would be deducted from any earnings of any kind whatever and not merely from earnings of collieries. This at any rate is the position in Britain, and if the same is true of the United States, I should not consider the Federal tax as affecting the profit; but, on the other hand, I should consider it as necessarily affecting the accumulative or gilt-edged rate at which the sinking fund is assumed to be invested. If a tax of 8 per cent. is payable on such investments, it is obvious that the sinking fund cannot accumulate at 6 per cent., but only at 5.52 per cent., and the fact that the valuer has not a 5.52 per cent. sinking fund table at his disposal is no excuse for using an incorrect rate; the calculation is a simple one. There are one or two interesting points that arise on such a problem, on which opinions are likely to differ; one of these is the method of dealing with working capital. Mr. Dilworth looks upon this as part of the total capital upon which interest is to be earned at the higher rate properly demanded for money invested in a speculative enterprise, or what I have called a risk rate, and I know many valuers who share this view. On the other hand, it may fairly be argued that if the working capital remains as flowing capital during the life of the mine, it is never exposed to risk of loss in the same way as money invested in shaft sinking, development, etc., and should, therefore, only be held to earn interest at the accumulative rate. I have no doubt, however, that this is a point upon which opinions will be divided.

Another point on which opinions differ is the proper method of dealing with development. My own method is to charge all developments until economic production

commences to capital; after this, the cost of development work is debited to a special development account, the account being credited with a payment of so much per ton of coal developed by the works in question, chargeable when the coal is actually won; in this way the cost of all such development becomes a portion of the cost per ton of coal winning. This is only one of several methods in current use. I agree with Mr. Dilworth that it is generally possible to obtain checks on a valuation from other considerations but that the true value is necessarily the capitalized present value of future earnings. I have usually found fairly close agreement among valuers as to the probable amount of coal in the ground, and even as to the probable future earnings per ton, the widest differences of opinion occurring usually on the question of what is the proper rate of risk interest to allow.

Requirements for Complete Face Mechanization in Coal Mining

BY R. Y. WILLIAMS,* POTTSVILLE, PA.

(Philadelphia Meeting, April, 1928)

IN the United States, fully 98 per cent. of the anthracite and bituminous coal tonnage obtained from underground operations is mined by the room-and-pillar system. Under this system, the total cost of the coal f. o. b. railroad cars contains as its greatest single item the money paid for "labor." Each labor dollar in an average bituminous mine may be subdivided into 85 c. paid for work done in by the gathering turnouts (including all work at the face), and 15 c. out by that point (including all work in and under the tippie). In an average anthracite mine, the corresponding figures are 70 c. and 30 c., the larger breaker force accounting mainly for the difference between these and the bituminous figures.

The next largest single item after "labor" is usually "supplies and power;" and a division of this item in by and out by the gathering turnouts shows somewhat the same relative proportions as has been noted under the case of "labor".

During the past 10 years, the chief advances made in the coal industry have been to obtain economies through complete mechanization covering that portion of the work which lies out by the gathering turnouts. These results have been accomplished by applying scientific principles to railroad yard layout, to cleaning and sizing machinery, to dumping facilities, to roller-bearing haulage equipment, and to heavy-rail, well-ballasted trackage. But these improvements, while gratifying in their resulting economies, apply only to 15 to 30 per cent. of the important cost items.

Obviously, then, the big problem before the engineers of the coal industry today has to do with the methods of mining and the layout in by the gathering turnouts.

Four years ago, Weston Dodson & Co., Inc., determined to have an exhaustive study made of the conditions which must be met in order to gain complete face mechanization, and in order that such a mining layout would be based on correct underlying principles. The result of that study, which was made by the writer, led to the adoption of a method of longwall mining wherein the roof is worked on the caving system and is supported as an overhang along the face by steel jacks.

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It is believed that some of the engineers who are vitally concerned with the task of mechanizing the basic face operations of coal mining will find interest in knowing the underlying reasoning on which we developed this method, particularly in view of the fact that we believe this method to be applicable to a wide range of mining conditions. Consequently this paper is presented under the following five main headings:

- A. Analysis of room-and-pillar method.
- B. Synthesis: the perfect longwall face.
- C. Observations on previous longwall trials.
- D. Basic principles of roof control.
- E. Results from use of steel jacks.

A. ANALYSIS OF ROOM-AND-PILLAR METHOD

The first step in our study was to make a careful analysis of the room-and-pillar method which is in such universal use in this country and which has over 100 years of history back of its development. The results of this analysis are set forth in the following seven subheads.

1. *Supervision and Inspection*

It would be interesting to have definite knowledge of the average number of working places in flat room-and-pillar mines where trackage, timbering, ventilation, etc., must be maintained at all times to provide means for obtaining a particular production. From partial data at hand, the number of places per 100 tons output per day varies greatly in different mines, but averages around 20 working places. A mine producing 1000 tons per day would therefore require 200 rooms.

According to the mine law in many states, all working places must be inspected by the foreman or one of his assistants at least twice per day during the time the miners are or should be at work. In gaseous mines the law requires an additional inspection prior to the time the miners enter their working faces. These pieces of legislation were enacted to help bring about safety; yet there are in addition a great many questions of procedure, cost, economy, etc., which should receive careful supervision to make mining successful.

Under room-and-pillar mining, the average daily inspection of working places consists of two 5-min. visits per face on the part of the foreman or one of his assistants. The supervisor is compelled because of the layout of the work to spend the greater part of his day in moving from place to place, with the result that an average working face has inspection during only 2.1 per cent. of the dayshift.

2. *Ventilation*

In order to provide for the safety and physical well-being of the miners, it is necessary to provide ample ventilation at the working places.

Good ventilation at the face maintains an atmosphere of normal oxygen content, carries off the products of combustion after shot-firing, and dilutes otherwise dangerous gases exuding from the strata.

With a good fan, it is an easy task to ventilate a dozen working faces; two dozen faces, though somewhat more difficult, can be easily arranged; but in the case of 200 separate and scattered faces for each 1000 tons of daily production, the problem of ventilation becomes a real one. Overcasts, doors, stoppings, regulators, brattices and curtains must be installed properly to course the air throughout the mine. All of these installations leak air, and, as the number of them gets greater and greater, the leakage becomes so enormous that the efficiency of the ventilation (*i. e.*, the percentage of the total entering air which reaches the faces) is low.

In room-and-pillar mines, there is no record of normal ventilation ever having blown off the hat of any miner working at the face; actually, it is difficult even in the best ventilated mines to "feel" any motion whatever in the air at the working faces. Yet in those same mines the main return to the fan may be traveling at a speed of 15 miles or more per hour; and in this connection, it should be remembered that the horsepower on the air varies as the cube of the velocity.

The room-and-pillar method, therefore, presents a complicated problem in the matter of ventilation; the efficiency of the fan is low, in that only a small portion of the air entering the fan ever reaches any working face in the mine; the installation of fan, stoppings, doors, etc., is high and is costly to maintain; and the operating cost for power is as high as the efficiency is low.

3. *Development*

The opening of an area in a mine for room-and-pillar mining consists usually in driving main haulage roads and air courses, secondary haulage roads and air courses from which rooms are turned, diagonal haulage roads between headings wherever required, crosscuts every say 60 ft. between headings and air courses for ventilation, room necks on say 45-ft. centers and 30 ft. deep along the secondary haulage roads, and break-throughs every 60 ft. between rooms to connect them for ventilation. All of this opening work is called "Development," and in practically all mines carries yardage rates which make the cost of development coal higher than the cost of room coal. In first mining, or in advance work, the percentage of the daily production from development work is 45, and from rooms proper is 55. If pillars are later robbed, and if the recovery is complete, the coal for a given area will be produced in the proportion, 25 per cent. from development and 75 per cent. from a combination of both room advance and pillar mining.

The reason for this added yardage rate (except in mines where the depth of roof or floor cut for height varies as between opening work and room work) is mainly due to the fact that development work is driven narrow, usually 9 to 12 ft., whereas rooms are driven wide, usually 18 to 24 ft.; and the money set up in the yardage rate is to compensate the development miner for the additional difficulty he must meet in winning only half as much coal between his two tight ribs as the room miner wins. In any mine where, say, \$2 per yd. is paid in development work over room work, the foreman, if you asked him, would say he paid no yardage in rooms. In one sense that answer would be misleading because the two tight corners in a room are exactly as tight as the two in a heading. What has happened in this case is that the work of the miner in cleaning out the two tight ribs of a room is included in the tonnage rate, and actually the yardage rate of \$2 in the heading is only part of the payment for tight coal, and represents the additional amount left over after the tonnage rate is paid. Shearing machines tend to equalize the actual loading cost per ton in rooms and headings; and, clearly in this case, the "yardage" (*i. e.*, shearing machine cost) in rooms is just half the "yardage" cost per ton in headings.

The room-and-pillar system, therefore, in the matter of development divides face work up into a great number of places, requires a rather large proportion of very narrow work on which yardage is paid, and actually can offer no face without tight corners. An honest analysis indicates that all mining under this system is narrow work.

4. Efficiency of Machine Performance

Stop-watch studies of average undercutting performance in room-and-pillar mines indicate an efficiency of only 20 per cent.; *i. e.*, the time spent in actual cutting is 20 per cent. of the operating shift; the balance, or 80 per cent. of the shift, is taken up in sliding the machine back from the face, in loading it on to the transfer truck, in transporting it to the next place to be cut, in unloading it from the transfer truck, in sliding it along the floor to one rib of the room, and in sumping it into the new face. Actually the two men known as the cutter and the scraper work harder during the 80 per cent. of the time when the machine is not cutting than they do during the 20 per cent. of the time when the machine is doing the task for which it was designed.

Concentration of working faces, improvement in transfer truck design, widening the room faces to the maximum the roof will permit, and perfect coordination of the work may show a percentage as high as 33 per cent. Surely any method of mining which at best can afford for undercutting machinery the opportunity to perform useful work during

only one-third of one shift per day, and then only in relatively flat beds of coal, cannot be the proper vehicle for complete mechanization.

Similar studies of average loading machine operations indicate an equally low percentage of performance; and in many cases the efficiency of the loading machine is still lower than that of the undercutting machine because of delay in spotting mine cars back of the loader. Inasmuch as most of the loading machine installations have been made in relatively thick coals, the machines have been able to show a profit; and temporary satisfaction with this partial profit has somewhat deadened the urge for condemning a method of low efficiency.

Gathering haulage equipment is also handicapped in the matter of performance. An elaborate network of tracks must be installed and maintained between the inby gathering partings and the 200 working faces per 1000 tons of daily production. Also a large number of haulage motors (or mules) and cars must be provided to make possible the "spotting" one at a time of three to six empty mine cars daily at each face and the gathering of these cars one at a time when loaded. This means 1200 to 2400 car-spotting operations per 1000 tons of daily production. Derailments, waiting for empties, and waiting for loads to be pulled, tend to reduce the loading capacity of the miners. In general, the gathering haulage problem is a complicated one, is beset with delays, and results each day in low round-trips per mine car and in low tonnages hauled per locomotive.

It will be found that all machinery designed for work at the face shows a low efficiency of performance under the room-and-pillar method because of the scattered nature of the mine layout.

5. Recovery in Tons per Foot-acre

Varying with the specific gravity of the coal, the total in-place tonnage of any bed averages 1800 to 2000 net tons per foot-acre. The lower figure might be considered as applicable to bituminous and the higher to anthracite coals.

The efficiency of any mining method in obtaining all of this in-place tonnage has a direct bearing on the cost per ton for such items as the original cost of the property, the cost of the installed plant, the total cost of development, and a large share of the cost of equipment. Charges to cover these items are usually set up in the cost statement as (a) "fixed charges" for depletion, depreciation, amortization, etc., and (b) development costs; and these charges vary inversely as the efficiency of the method of mining.

The "fixed charges" are determined by the accounting department on the basis of the total tonnage in the property reduced by the expected inefficiency of the mining method, and are usually so definitely "fixed"

that a superintendent who increases the efficiency of the mining method can see no advantage thereof except in the reduction of development costs.

To illustrate the importance of high efficiency of recovery in the mining method: Let us assume that the proper charges for these items of property, plant, development, and equipment would be \$1 per ton under 100 per cent. recovery. If the actual recovery under the mining method were only 70 per cent., the \$1 divided by 70 per cent. would be \$1.43 per ton. Forty-three cents per ton is a tremendous loss, due solely to the inefficiency of the mining method.

The results of room-and-pillar mining throughout the United States indicate that recovery is seldom complete and that never can all of the coal be recovered mechanically. Entire states, like Illinois, produce only 50 per cent. of the original bed content; the steeply pitching beds of the Southern anthracite probably do not exceed 60 per cent.; and even in those fields where a high recovery is obtained, the cost of the last tons is high because "pick rate" must be paid, additional timber must be set, and the roof proves difficult to control.

6. *Quality of the Product*

There are two kinds of quality which are affected by the mining method; namely, (a) size and (b) purity.

(a) *Size*.—In the room-and-pillar method, with 200 faces per 1000 tons daily production, each face that is to be worked on any particular day presents its coal locked in by floor, roof, back, and two ribs—five out of the six sides are tight. It is the general practice in bituminous mines, but is seldom done in anthracite rooms, to undercut, middlecut, or overcut the face before shooting. This reduces the number of tight sides to four; but as has been previously noted the operation of the undercutting machine is inefficient in that it cannot work continuously. In a few soft-coal mines, machines are used to shear the coal on the two ribs in addition to undercutting. This reduces the number of tight sides to two; but here again a machine is introduced which operates under the same low efficiency as the undercutting machine because it cannot work continuously, and in addition it makes slack of the shearings. In general, then, the room-and-pillar method presents its coal in a multitude of places, each place being narrow and having the coal locked in on four or five sides.

The universal method for unlocking this tightly held coal is to use explosives. Theoretically, the quantity of explosive required per ton of production depends solely on the tightness of this "lock." But the miner is far more interested in the quantity than he is in the quality of his production, and he tends invariably to use more explosive than actually needed. Many companies have made exhaustive tests to determine the best methods for shooting the face, featuring in their studies such ques-

tions as: depth and angle of each hole, placement and sequence of firing of the several holes, kind and weight of explosive per hole, kind and character of tamping, etc. All of these studies are for the purpose of obtaining the maximum lump coal; but of what avail is it to know the intricate details of the best possible method of using explosives if, through inefficient supervision, it is almost impossible to get miners to follow it?

In the anthracite the difference between the values of prepared and junior sizes is about \$6 per ton; and in the bituminous the difference between the values of lump and slack is about \$1.50 per ton. As important as these values show the size of product to be, the room-and-pillar method cannot hope to afford a means for the highest possible lump percentage because the coal is locked so tightly in narrow working places, and because the lack of close supervision permits the largely uncurbed use of explosives.

(b) *Purity*.—The room-and-pillar method (except in the steeply pitching beds of the Southern Pennsylvania anthracite) is founded on the principle that clean coal will be produced in each working face by each miner. This is so firmly believed that many officials would refuse even to consider the installation of loading machines because (as they say) all the impurities in the bed would then be loaded with the coal into the mine car.

Let us analyze this matter to determine the actual facts. In one-bed mines, or in a particular bed of multi-bed mines, it is the usual custom to pay the room miner a definite rate per ton or per car for the coal which he loads. If a foreman, who might be in the act of disciplining a miner for loading dirty coal, were asked how much he paid a miner for cleaning and gobbing any impurities found within or above the bed, he would reply, "Nothing: the miner according to contract is supposed to load clean coal."

The first word of this foreman's answer is wrong. While it is true that he pays nothing *in addition* to the tonnage or car rate, that same rate includes all work performed by the miner in producing coal; such as, cleaning the coal, gobbing of normal draw slate, timbering, drilling, shooting, loading, laying track, illuminating the face, waiting for cars, etc. The only way to determine the proportion of the mining rate applicable to any of these various duties of the miner is by studying them with a stop watch. Such a study would show that in some beds the actual cost of cleaning at the face is high, and in addition all the impurities which the miner carelessly or willfully loads with the coal is paid for as coal.

The balance of the foreman's answer is correct in that the coal industry has in the past always followed the practice of paying the high-priced miners for cleaning the coal in the dark at the face instead of paying slate-pickers less hours at half the wage rate to clean the coal under

daylight conditions on the surface. That the attempt to depend solely on the miners for the purity of the coal shipped to market has not succeeded in practice is proved by the increasing use of slate-pickers to remove the slate which the miners load with the coal.

On the basis of hand-cleaning, it would be cheaper to clean solely in the preparation building rather than to attempt to clean in the many underground working places, *provided*, the rate paid the miner per ton or per car could be reduced in the exact proportion shown by the results of the stop-watch study of his activities. If this is true for hand-cleaning, the savings under mechanical cleaning would be large and would offer the further advantage of uniformity of product.

Therefore, the room-and-pillar method must be considered inefficient in the matter of purity of product, because: the miners, who receive the highest wages of any class of workmen at the mine, are paid to produce clean coal; these miners, on account of lack of supervision, load impurities and are paid therefor equal or higher rates than they are paid for coal; and, because of the failure of this system, additional cleaning facilities must be provided in the tipple or breaker.

7. Class of Work

There are three methods of paying miners for work accomplished: (a) by piece work, (b) by day labor, and (c) by task work.

(a) *By Piece Work*.—Under this method, tons are weighed, cars are counted, or yards are measured. This method is liked by the ambitious miner; but is provocative of high cost because (1) the piece work rate is apt to be based on a living wage for the poorest worker; (2) the tendency of the Union seems to have been to curtail the production of the best miners; (3) supervision is usually so poor that little instruction can be given to the poor workman, who, when he fails, claims make-up wages; and (4) any improvement to assist the efficiency of the miner, such as shearing the ribs, blasting with shotfirers, etc., is paid for entirely by the company, the previous contract rates remaining despite the reduction in the work of the contractor.

(b) *By Day Labor*.—Under this method, shifts are paid for on the basis of hours worked and without regard to accomplishment. This method is liked by the loafer who makes an easy job of it because of the lack of supervision. It is not liked by the ambitious miner who will not long continue efficient when he sees about him inefficient men drawing pay equal to his own.

(c) *By Task Work*.—Under this method, tons, yards, cars, hours and all items of work are assembled as data to determine the average number of honest man-days required to perform a daily group task. When the daily task is completed, each member of the group receives his

share of the total pay without regard to the particular tonnage loaded, the hours worked, etc. Of course, this task work requires continuous supervision; but when supervision is continuous it can fulfill its major function of service to the group rather than of mere criticism of the individual. This method of paying for work at the face seems very popular with the men, who like to work in groups and who give valuable assistance to supervision in recommending good workmen for their group and in preventing absenteeism.

The room-and-pillar method almost universally uses piece work as the basis of paying for work at the face; it uses day labor for paying for work outby the face; but it is preempted from the possibility of using the task work basis because of the tremendously scattered character of the mining layout.

Summary

Our analysis of the room-and-pillar method has shown: that it is woefully weak in the matter of supervision; that it is difficult to ventilate; that it requires heavy development expenditure; that it is unsuited for the introduction of machinery for cutting, for loading, and for gathering haulage; that it does not afford an opportunity for the economic recovery of the total coal in the bed; that the quality of the coal produced is apt to be poor both as to size and purity; and that the working conditions are such as to require high rates for low performance.

B. SYNTHESIS: THE PERFECT LONGWALL FACE

Because of the defects found to exist in all room-and-pillar methods, many experiments in lengthening the faces have been made with greatly varying and often indifferent results. The main criticism which can be made of these experiments is that most of them lack any careful consideration of the goal toward which they are aiming. Should not this goal be definitely visualized before any experimentation is started, and should not there be postulated a perfect longwall face before money is spent to determine whether or not perfection is possible? Inasmuch as we believed our goal should be definitely visualized, we determined as the second step in our study to build up by synthesis a theoretical longwall which would fulfill ideally the requirements of complete face mechanization, and then to test this longwall by the same kind of analysis as we applied to the room-and-pillar.

All considerations of perfection in the longwall method of mining point to the following requirements of each face:

First Requirement.—Each face must be just long enough for an undercutting machine operating continuously to cut it in one shift of 8

hr. This length would vary from 300 to 500 ft., depending on the severity of the cutting.

Second Requirement.—Each face must be capable of being undercut to just such a depth that a loading machine operating continuously can load the coal in one shift of 8 hr. Such a cut will average $6\frac{1}{2}$ feet.

Third Requirement.—Each face must be capable of being held open back from the face for a width sufficient to accommodate the roof-control mechanism, to permit the operation of the cutting machine, and to provide space for a conveyor of such capacity as will convey all the coal from the face to trains of mine cars in one shift of 8 hr. The width of this space back from the face will average $10\frac{1}{2}$ feet.

Fourth Requirement.—Each face must be capable of having the mechanism of roof-control completed daily in one shift of 8 hr., but must also be capable of standing indefinitely without requiring attention during an idle period.

A perfect longwall face, such as described above, would be capable of performing one complete cycle every working day of 24 hr., and would permit the carrying out of three major operations, namely, loading, cutting and roof-control, and drilling and shooting, on separate shifts of 8 hr. each.

An analytical test of the efficiency of such longwall mining is given in the following comparison with room-and-pillar.

1. *Supervision and Inspection*

Instead of having a method which like the room-and-pillar requires on the average the maintenance of 200 rooms for a daily production of 1000 tons, the perfect longwall would require on the average only three faces. Instead, therefore, of inspection being only 10 min. per 8-hr. shift as under room-and-pillar, there would be continuous inspection under longwall.

2. *Ventilation*

In the matter of ventilation, the longwall method shows the same simplification over room-and-pillar as has been noted in the matter of supervision. In an actual case, the design of the fan to ventilate room-and-pillar workings in a particular mine called for a 225-hp. motor, whereas the design of the fan to ventilate an area of longwall workings to yield the same tonnage called for only a 25-hp. motor, and the quantity of air sweeping the longwall faces was estimated as four times greater than that sweeping the room-and-pillar faces.

3. *Development*

As will be noted later in this paper, it is our recommendation that all haulage roads for longwall mining be maintained by means of solid ribs

of coal rather than by means of pack walls in the gob. It is therefore necessary to drive main haulage roads and air courses, secondary haulage roads and air courses, and the necessary diagonals and crosscuts in much the same manner as has been the practice under room-and-pillar. However, there are two points which show greatly to the advantage of longwall over room-and-pillar in this matter of development: (a) Under longwall, the proportion of total production coming from narrow headings and air courses varies from 6 to 12 per cent., whereas under room-and-pillar the corresponding proportion varies from 25 to 45 per cent. and (b) under longwall, the remaining 88 to 94 per cent. of the production comes from faces 300 to 500 ft. long without any rib anywhere, whereas under room-and-pillar the remaining 55 to 75 per cent. comes from still very narrow room faces (usually not over 24 ft.) and each face locked in by two ribs.

4. Efficiency of Machine Performance

Whereas room-and-pillar has shown an average of 20 per cent. and a maximum of 33 per cent. as its efficiency in providing continuous operating opportunities for cutting, loading and haulage machinery, the perfect longwall face should offer over 80 per cent. efficiency. In other words, longwall should for the same operating cost produce three times the output per machine. Or, as a corollary, complete mechanization covering cutting, loading and gathering haulage, could be installed under longwall for one-third the cost of that under room-and-pillar for a given tonnage.

5. Recovery in Tons per Foot-Acre

The longwall method permits the complete recovery of all the coal desired from any bed. In this respect it is superior to the room-and-pillar method which seldom obtains complete recovery, and then only at a high cost for the "last" tons. Therefore, under longwall the cost per ton for fixed charges and development will in general be much lower than under room-and-pillar because of this ability to recover all the coal.

6. Quality of the Product

(a) *Size*.—Under longwall, the long faces which contain 88 to 94 per cent. of the coal for market present that coal locked in on only three sides—floor, back and roof—as compared with the five sides of room-and-pillar. Furthermore, these three sides can be reduced to two sides by a cutting machine which is able to work continuously and therefore efficiently. The long faces, 300 to 500 ft. long, with the coal undercut, overcut, or middle-cut, present the coal in an ideal form for blasting with the minimum of explosives. The absence of any necessity for heavy rib shots to clean out tight corners greatly reduces the breakage caused by

explosives. It is believed that in a great many coals, long faces without ribs would permit the use of a hydraulic cartridge or some similar mechanism which would entirely dispense with powder. And because supervision under longwall is efficient, whatever procedure in blasting may have been shown by test to be best adapted to the production of lump on any face may be assured of being used in practice. Thus it appears that longwall offers the best possible opportunity to obtain the maximum of prepared over junior sizes in the anthracite and lump over slack in the bituminous coals. Actual recent results have shown in anthracite that the machine cuttings alone contain 50 per cent. prepared, and that the balance of the coal shows much higher prepared than in room-and-pillar; and in bituminous there has been at one mine an appreciation in the value of the product of 22 c. due to an increase of $14\frac{2}{3}$ per cent. in the proportion of lump over the former room-and-pillar.

(b) *Purity*.—It is difficult to make a general statement to cover the results that may be expected from complete mechanical longwall in the matter of purity. Each mine must be studied as a separate unit to determine the effect upon it of the introduction of mechanical longwall.

Mention has previously been made of officials who refused even to consider the installation of loading machines because, as they said, "All the impurities would then be loaded out with the coal." As a matter of fact, one of these officials was a superintendent operating a mine in the Pittsburgh bed which because of its great extent and high quality is one of the greatest producers in the United States. Actually this superintendent was wrong in his reasoning for refusing to consider a loading machine. At his mine the coal was 6 ft. thick and carried two sources of impurities; (a) two "bearing in bands" which occupied a 4-in. space 20 in. above the floor, and (b) draw slate which averaged 10 in. in thickness, which lay immediately above the coal, and which came with the coal during blasting. Tests on long faces under identical conditions have shown the economical possibility of removing these sources of impurities mechanically at the face, (a) by middle-cutting the "bearing in bands" at a cost of 2 c. per ton of production, and (b) by forepoling the draw slate at a cost less than the cost of letting it come with the coal. These results show that in this particular mine, which is important in that it represents many like mines, longwall presents a great advantage over room-and-pillar in that it provides a means for sending from the face absolutely clean coal requiring no preparation in the tippie other than sizing; and the entire operation is mechanized.

It is freely admitted that not all beds are capable of being completely cleaned mechanically on long faces. However, it has been previously pointed out under the room-and-pillar method (a) that cleaning at the face is expensive because of the high rates paid to miners, (b) that miners do not load clean coal, necessitating both paying the high mining rate

on the tonnage of refuse loaded and paying slate-pickers for removing the impurities at the tippie, and (c) that mechanical cleaning is required to replace slatepickers in order to prepare a product for critical markets. Therefore it seems possible to state that a consideration of all the facts in the case does not admit that the question of purity of product can be used as an argument against the introduction of the longwall method.

7. *Class of Work*

Long faces completely equipped with cutting, drilling, loading and conveying machinery, all working continuously and efficiently, offer an ideal setting for the task-work system of paying for the work done. Our experience indicates that workmen like laboring in groups, do best when their task is definitely arranged and placed before them, appreciate supervision when they see it is helpful rather than critical, enjoy the fact that longwall faces are electrically lighted throughout, and by experience eventually come to an appreciation of the greater safety from roof falls and from mine-car accidents.

Summary

Our analysis of this postulated perfect longwall method has shown that, if we can devise a means of roof-control which can in one shift so handle the roof as to permit the operation of a face of definite length and depth of undercut, the advantages over the room-and-pillar method will be tremendous. These advantages are: continuous inspection; adequate and economic ventilation; low development costs; opportunity for the efficient use of cutting, loading and gathering haulage equipment; total coal recovery; excellent quality as to size and purity of production; and improved working conditions under reasonable rates and high performance.

C. OBSERVATIONS ON PREVIOUS LONGWALL TRIALS

In order to gain the advantages which attached to our theoretically perfect longwall, it was necessary to find or develop an adequate method of roof-control. Consequently the third step in our study to obtain complete face mechanization was an investigation of the then existing attempts to operate long faces in the coal fields of the eastern part of the United States. This investigation consisted of personal underground visits, occupied a period of several months, and was as keenly interested in failures as it was in successes of roof-control methods.

In this paper, no description of the mining layout or the mechanical equipment used at the visited mines will be given; first, because that would be a duplication of the very able work of G. B. Southward, whose valuable reports are appearing in the current issues of the *Mining Congress*

Journal; and, second, because our study was not particularly concerned with either the layout or the equipment, inasmuch as we had postulated a perfect longwall layout with definite dimensions as to length of face, depth of cut, and width to be kept open for the cutting, the loading, and the conveying machinery. The sole and vital interest of our study was to find or develop a method of roof-control which would permit us to put into practice our theoretically perfect faces.

Consequently our findings in that investigation will be given in this paper merely as general observations in the matter of roof-control.

1. *Pack-wall versus Caving Systems*

There are two methods of longwall roof-control, known as (a) pack-walls and (b) caving systems.

(a) *Pack-walls*.—This method consists in using stone obtained from the bed, the roof, the floor, or brought into the mine from the surface, in order to build walls along and close to the face leaving open along the face a narrow space in which the miners work. In longwall advancing, additional walls in pairs are built at approximately right angles to the face and on about 30-ft. centers to act as artificial ribs to hold open haulage roads leading back from the face through the gob.

The advantages of this method of roof-control are: no explosives are required, because the settling of the roof breaks down the coal; the resulting product is of good size; all the coal in tons per foot-acre is recovered; damage to the surface is minimized; and in the longwall advancing there is no development required. However, the disadvantages which more than offset these advantages are: most of the work of securing the material for the pack-walls and all of the tedious work in building these walls must be done by hand and cannot be done by machines; it is usually adaptable only to thin beds; it operates best only under particular kinds of shale roof; the width which can be kept open between the face and the walls is so small as barely to accommodate the undercutting machine; because of this lack of width it is difficult to use a conveyor and still more difficult to use a loading machine; the maximum depth of undercut is small, not over 3 to 4 ft.; the tonnage per day per foot of face is small; the labor cost per ton is high; and there is a large item of cost in keeping the haulageways "brushed," in order that as the roof settles the roads may be kept open for the passage of cars.

Although the pack-wall method is in almost universal use abroad, possibly due in large measure to the question of surface damages, it has found little favor in the United States. High cost, low tonnage per employee, and unsuitability for complete mechanization ruled it out in our search for a method of controlling the roof along our proposed longwall faces.

(b) *Caving Systems*.—Caving systems are based on the plan of placing artificial supports along the face and at a sufficient distance from it to provide space for men and machines to operate, and of removing these supports as soon as the face has advanced and new supports have been installed. It is expected that, as soon as any supports are removed, the roof previously held by these supports will cave and thus relieve the pressure on the face.

In all of the mines investigated, this caving system was being used; and, also in every case, timber was the material used for the artificial supports. In theory, this system seemed to offer a means of roof-control that might fill all the requirements for the operation and maintenance of our perfect long faces; but in practice, it was found that the daily results were not uniform, and that the faces were frequently lost through premature roof caving. Consequently the second observation in our study of roof-control covers the adequacy of timber for the artificial supports to be used in caving systems.

2. *Timber as Artificial Supports*

The investigation showed that timber for artificial supports in caving systems was being used in two ways; either (a) as props, or (b) as log-cabin or solid cogs.

(a) *Props*.—The props were 8 to 10 in. dia., were set close together, and were placed in 4 to 6 rows parallel with the face. In general, two new rows were installed after each cut of coal had been loaded out, and then the two rear rows were removed. During the first 40 to 60 ft. advance of a new face, the props to be removed were dislodged by means of post-pullers; but thereafter, as weight came on the props, the removals had to be accomplished either by chopping (which is contrary to the mine law in most states, and is certainly not desirable) or by blasting. Naturally these props could not be reused, and the timber consumption per ton was usually more than double that under room-and-pillar mining. Roof rides were a continual menace, and faces were frequently lost.

(b) *Cogs*.—Cogs were built usually of 6 by 6-in. hardwood with a length varying from 24 to 42 in. In general, it was noted that log-cabin cogs were used in initial attempts at long face work, whereas in later work solid cogs were in use. In most cases, the cogs were built on a base of 4 to 6 in. of slack coal which was to assist in the subsequent recovery of the cogs.

The use of cogs showed an advantage over props in that the ultimate timber consumption was less; but it was still higher than under room-and-pillar because individual sticks of each cog would be lost on account of being crushed by roof pressure or being buried by caves.

Cogs however showed to a disadvantage in comparison with props in the matter of being easily compressible. Because of this tendency to yield, the roof weight kept continually creeping over the supports toward the face, thereby reducing the depth to which the coal could be safely undercut and causing the likelihood of the immediate roof breaking and falling in slabs throughout the area wherein the men and machinery were operating.

Inasmuch as long faces were frequently lost through the failure both of props and of cogs, superintendents were tending to change the layout in the hope that they could in this manner find a way of safeguarding the control of roof. These tendencies were as follows:

3. Tendencies in Long Face Layout

The first tendency was to adopt a long face layout wherein the face would be stopped after an advance to a point slightly less than the distance which experience in that particular mine had showed roof failures were liable to occur (usually less than 100 ft.); the equipment would be removed; a pillar 15 to 30 ft. in width would be left along the entire length of face; and the long face would start again on the other side of this pillar in a specially prepared narrow room driven for the purpose. This layout caused a high charge for the following: for man-hours to remove and reinstall the equipment, for development mining to open the narrow room, for interruption to production caused by this delay, and for the loss of the tonnage remaining behind in the pillar.

The second tendency on the part of superintendents was to limit the length of the long faces in the endeavor to overcome the dangers of roof collapse. The men who favored this tendency argued that if they could safely advance a 400-ft. face somewhat less than 100 ft., they could just as safely advance an indefinite distance with a face whose length was somewhat less than 100 ft. They argued, further, that this would reduce, in comparison with the first tendency, the man-hours for the frequent removal and reinstallation of the face equipment, the interruption to production, and the loss of coal left behind in the pillar. However, in comparison with our ideas of a perfect longwall, it had the disadvantage of greatly limiting the efficiency of machinery, of ventilation, of supervision, and of more than doubling the total amount of the required development.

The third tendency on the part of superintendents to alter the layout was to adopt the so-called "step system" wherein several faces, each usually somewhat less than 100 ft. in length, were driven simultaneously in retreat, and each face was "stepped" somewhat less than 100 ft. ahead of the neighboring face on one end and an equal distance behind the neighboring face on the other end. Delineation of a group of five

such faces on the mine map would resemble a flight of five steps on which each tread would represent a working face and each riser would represent the development heading to accommodate a train of empty mine cars for the loading of the coal from one of the faces. This layout is probably the most efficient plan of operation for use when timber in the form of props or cogs is to act as the artificial support of a caving system. Let us therefore examine it to see how nearly it comes to affording the advantages which we are seeking in our perfect long faces.

In the first place, there would be required five 80-ft. faces to equal one 400-ft. longwall face. The disadvantages of this reduction in the length of faces would be: in the loss of efficiency for cutting, loading, and gathering haulage machinery; in the need for five conveyor drives instead of one, affecting both the first cost or purchase price and the operating cost covering the daily move forward and the anchoring of the drives; and in the reduction of efficiency for supervision and ventilation due to the increase in the number of places to be served.

In the second place, the depth of cut on the step faces could not safely be made $6\frac{1}{2}$ ft. as desired in our assumed perfect faces. This failure to be able to make a deep cut is due to the concentration of pressure exerted by the roof near that end of the face which approaches the area mined out by the adjacent face. The disadvantage of this lack of depth in the undercut would be the still further reduction in the efficiency of the loading, conveying and gathering haulage machinery.

In the third place, the total development required by the step face layout would be more than double that required by our assumed perfect longwall faces. This would be to the disadvantage of the step face not only in this necessity for double development but also in the fact that all of the development for a particular panel would have to be completed before any long face could start, thus freezing working capital.

And, in the fourth place, the five points where conveyors load the face coal into mine cars are not on the main longwall heading but are up on the five additional headings driven for this purpose. Consequently, switches, frogs and trackage must be maintained in each of these headings; this equipment must be recovered as the faces retreat; cogs must be maintained on one side of each heading to hold open a haulageway from the loading point to the end of the "step" of the adjacent face; five workmen instead of one are required to have charge of the loading of the cars; and the gathering haulage equipment is handicapped in having to place short trips on stub tracks instead of being able to maintain a steady stream of cars by means of long trips on a straight track.

We were quick to recognize the advantages which the step face layout offered in comparison with the room-and-pillar; but these advantages fell far short of those we had hoped to obtain through our preconceived long face layout.

The general conclusion we reached in our investigation of these early trials of long faces was that the one and important task before us was to find a substitute for timber as the artificial support for the caving system. But before starting experimentation along this line, we determined as the fourth step in our study to set forth such data as we had on the principles of roof-control.

D. BASIC PRINCIPLES OF ROOF-CONTROL

Unfortunately there is available very little scientific information to help us in understanding the many variables in roof stresses. However, let us examine such practical data as may be at hand.

1. *Roof Action in Room-and-pillar Mining*

It is easy to figure accurately the total load overlying a given area. This is found by multiplying the area in feet by the height in feet by the weight of a cubic foot of the roof material. Thus, a room 24 ft. wide, 300 ft. long, and 600 ft. below the surface would have a total load overlying the excavation amounting to

$$24 \times 300 \times 600 \times 160 = 720,000,000 \text{ lb.}$$

Clearly this load does not equal the working pressure on the face. If the roof were fairly good and the coal were 6 ft. thick in the room over which we have just figured the total load, the timbering would probably consist of three rows of 6-in. props on 5-ft. centers, making a total of 180 props. The safe load which each prop would carry would be about 14,000 lb., so that the 180 props would carry safely a load of 2,520,000 lb. However, the total capacity of all the props in the room would be only one-third of 1 per cent. of the total load over the excavation. In other words, the installed timber would carry safely only 2 ft. of roof material if that material were perfectly free to fall. The function of timber, therefore, has practically nothing to do with the support of the roof; but it is confined to holding in place any slab which might tend to get loose, and to giving warning of approaching roof settlement.

If it is not the timber, then what is it that holds the roof intact over a room in room-and-pillar mining? Clearly the strength of a good roof must be an inherent function of the successive strata immediately overlying the coal and acting as beams spanning any opening from rib to rib. If this is true, then the same dimensions of rooms under similar immediate roof conditions could be used without any consideration being given to the depth of the cover.

Immediate exception may be taken to the above conclusion. Engineers who have laid out room-and-pillar work under both shallow and heavy covers will state correctly that the same dimensions of rooms and pillars cannot be used without reference to depth of cover. But, remem-

ber, our conclusion was made only with respect to *room* dimensions. To analyze this condition: Under relatively shallow cover, the room-and-pillar layout usually removes 50 per cent. of the coal in advance work. When a panel of rooms has been first-mined, the total overlying load carried by the coal pillars between these rooms has been doubled. If difficulty develops just prior to pillar extraction, this difficulty will show itself in one of three ways: first, the coal pillars may crush if they have not sufficient strength to carry the doubled load, and the resulting condition is called a squeeze; second, the stratum usually consisting of fireclay lying immediately beneath the coal bed may not have sufficient strength to carry the doubled load and will slowly flow out from beneath the pillars, causing what is called a heave; or, third, a stratum of weak material, like a soft fireclay, lying just above the immediate roof may under the doubled load succeed in finding an outlet by breaking the shell of immediate roof near the center of the rooms and may induce what is termed a series of caves. It appears therefore that for increasing depth of cover, the necessary changes in dimensions of rooms and pillars are due to the requirement of leaving a large proportion of coal during first mining, a requirement which can be met equally by increasing the pillar width or by decreasing the room width. The reason the pillar width cannot be indefinitely increased is because difficulties would then develop in the extraction of the pillars.

We may therefore summarize roof action under room-and-pillar mining as being a uniformly loaded beam rigidly supported by the two ribs of coal; and that excellence of a roof depends (a) on the inherent strength of the first few feet of strata immediately overlying the coal; and (b) on the success of the layout in preventing any general roof movement through the use of ample pillars.

2. *Roof Action in Longwall Mining*

It is a well-known fact that, in any large fall or cave in flat workings, a vertical section at right angles to the horizontal breakline will show an upward break on a line roughly at 45° to the strata and pointing away from the rib. In all of the longwall faces which have failed on account of the roof riding over the supports, the writer has yet to see or hear of a case where it was not possible (though difficult and costly) to recover for re-use most of the equipment and machinery on that face. After all such failures, there is generally ample room for a man to crawl along the face hugging tight to the coal rib. Such observations tend to prove that the working pressure of the roof on a longwall face is much less than the total load, just as was found to be the case in room-and-pillar mining. If the total load were able to exert its full weight on the face, the pressure would be so great that recovery of equipment would be out of the

question, and even if recovered the equipment would have been wholly destroyed.

What function of roof action, then, tends thus to hold open a space next to the face of a longwall? It is the cantilever action of each roof stratum overlying the bed of coal. The first foot or two of immediate roof above the coal extends out a foot or two beyond the face; the next foot or two above the first layer extends out beyond the face a foot or two farther; and so on to establish a roughly 45° line of break extending upwards as far as we can see.

It is probable, however, that this 45° breakline does not extend all the way to the surface. It has been observed in frequent instances of longwall that, whereas all surface subsidences lag the advance of the long face, the surface breaks often precede the face, occasionally many feet in advance of it. To explain these findings: It is believed that this 45° breakline extends upward from the face to a point where a particular stratum is prevented from breaking and falling into the pile of caved material because of there being no space for it to occupy. This is due to the fact that caved material occupies a space $1\frac{1}{2}$ to $1\frac{3}{4}$ times the space it occupied in the solid. The several strata lying beneath the particular stratum we have in mind have so swelled in breaking that they have filled the space for caving provided by the extraction of the coal. Our particular stratum, therefore, is for a time partially supported by the material in the cave. It is carrying, however, all the load of the material between it and the surface; and this load becomes greater and greater as the longwall advances. The caved material which is partially supporting it tends to contract through its own weight and through the great overload caused by the bending and sagging of the particular stratum. This latter stratum continues to sag until there is a rupture—but this rupture does not take the form of a cave; instead it opens up at a natural roughly vertical fissure and pulls horizontally along its bedding plane. The several strata above our particular stratum continue to ride on it and to rupture in the same manner, each stratum tending to open at a fissure ahead of the opened fissure of the stratum below it. In this way a new fracture line is set up which may be at roughly 45° to the bedding planes, but which points back toward the face.

We have thus shown from empirical data how roof action on longwall faces tends to hold open through the cantilever action of each roof stratum a small area in front of the face; how the roof beyond the ends of those several cantilevers tends to break off and cave; and how this roof behavior continues from the coal face up toward the surface to an unknown point above which the total load has no effect on the face, this total load on one side of the breakline being carried by the material in the cave, and on the other side being carried normally by the undisturbed strata back over the coal.

Our problem therefore is to hold open a space of definite length and width in front of the longwall face, this width being greater than that which the roof action can accomplish unassisted. In other words, the artificial supports which we are to install must be able to transfer the point where cantilever action of roof strata starts to a point away from the face at a distance therefrom equal to the width of the space we desire to keep open. The roof over this working space will then be a beam having one end resting on the solid coal and the other end resting on the artificial supports. This beam is like the beam spanning the two ribs of a room in the room-and-pillar system except that in longwall one rib has been removed and artificial supports have been substituted.

The load on this beam theoretically will be carried one-half by the solid coal and one-half by the artificial supports; and the weight of this load may equal the total load of the strata lying between the coal and the unknown point where the breakline changes its direction.

It is thus evident that the artificial supports have a very different function in longwall from what we found to be their function in room-and-pillar. In the latter case, the props merely steadied the immediate stratum of roof overlying the coal and gave warning if the coal pillars were too small; but in longwall a part at least of the total load must be actually supported. Does not this analysis show the reason why timber either in the form of props or cogs is proving inadequate to support safely the desired working area along a longwall face? An example might show this more clearly, even though some of the variables are unknown:

Let us assume a longwall face 400 ft. long which we wish to keep open a total width of 17 ft. (including the $6\frac{1}{2}$ -ft. undercut). Let us further assume that we can devote $3\frac{1}{2}$ ft. of the 17 ft. of width to the space for our artificial supports, and that the unknown point up to which we must provide support is 100 ft. above the coal bed. Then the load to be carried by the artificial supports will be

$$\frac{1}{2} \times 400 \times 17 \times 100 \times 160 = 54,400,000 \text{ lb.}$$

To support this load safely would require 3900 6-in. props or 1063 12-in. props if recourse must be had to timber, and all of these props would have to be accommodated in an area $3\frac{1}{2} \times 400 = 1400$ square feet.

Our analysis indicates that the artificial supports to hold open safely and economically a space 17 ft. back from the face of a perfect longwall must have three characteristics: first, *strength* to carry the loads set forth in the above example; second, *rigidity* to induce caving immediately beyond the supports and to hold the roof beam over the working space intact; and third, *movability* to facilitate the daily advance of these supports with safety both to the men and to the supports.

Timber either as props or as cogs does not fulfill any of these three requirements. Therefore recourse must be taken to the development

of steel jacks. A jack designed to carry safely 200 tons would equal the carrying capacity of twenty-nine 6-in. props or eight 12-in. props; it could be made to have as complete rigidity as required; and it could be equipped with tripper device which would easily accomplish its collapse with safety to the men.

Actually, the next step in our study was a series of experiments to develop a suitable design of steel jack. In this paper, however, it seems hardly necessary to describe these experiments except to state that we recognized the possibilities of various types of jacks; such as hydraulic jacks, sand jacks, wedge jacks, screw jacks with steeply pitched threads, and any combination of two or more of these types. The jack which finally won temporary approval and by means of which most of our work has been done was a wedge jack designed by us and manufactured by the Lorain Steel Co., Johnstown, Pa.

The last step in our study was to put our ideas and theories into practice by submitting them to the actual test of underground operation. Some of the results obtained are listed in the following paragraphs.

E. RESULTS FROM USE OF STEEL JACKS

Nine longwall faces in four different mines have been driven a total distance of over one mile. Of these nine faces, four completed the extraction of all the coal in a predetermined area and permitted the removal of all the equipment to new locations; one is temporarily idle, and four are still advancing daily. The maximum advance of any one face to date has been 2400 ft., and that face has 1000 ft. more to go to reach the property line.

From the experience gained in this work, we may set down the following observations:

1. Arrangement of Jacks

(a) *Immediate Roof Strong.*—If the immediate roof is strong, the jacks are arranged as shown in Fig. 1. The double outer row of jacks is called the break row and is spaced on 4-ft. centers. The single inner row of jacks is called the safety row and is spaced on 8-ft. centers. A complete cycle of operation, starting with the conditions shown in Fig. 1, would be to blast the coal, to load it into the conveyor, to collapse the safety row and reset it within 4 ft. of the new face, to advance the conveyor $6\frac{1}{2}$ ft. with the aid of post-pullers, to install a new double break row of jacks, to collapse the old double break row and permit the jacks and timber to lie idle between the new double break row, and to undercut the face.

(b) *Immediate Roof Only Fair.*—If the immediate roof is only fair, the jacks are arranged as shown in Fig. 2, in order to obviate the necessity for collapsing the safety row. The double break row is set as in the first

case on 4-ft. centers, but the single safety row is set on 4-ft. centers instead of on 8-ft. centers. A complete cycle of operation, starting with the conditions shown in Fig. 2, would be to set an additional row of safety jacks under the inner end of the collars previously held only by the first safety row, thus making the double safety row an exact duplicate of the double break row; to blast the coal; to load it into the conveyor; to disjoint the conveyor and reassemble it immediately inside of the double safety row; to collapse the double break row, permitting one row to lie idle between the jacks of the new break row and using

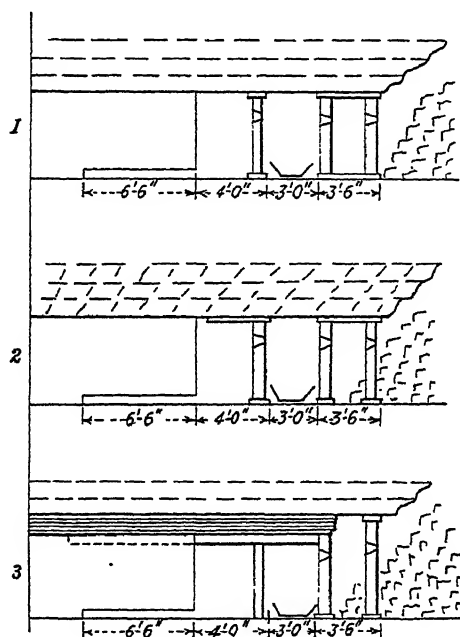


FIG. 1.—ARRANGEMENT OF JACKS WHEN IMMEDIATE ROOF IS STRONG.
 FIG. 2.—ARRANGEMENT OF JACKS WHEN IMMEDIATE ROOF IS ONLY FAIR.
 FIG. 3.—ARRANGEMENT OF JACKS WHEN IMMEDIATE ROOF IS BAD.

one row to form the initial row of the new double safety row; and to undercut the face.

(c) *Immediate Roof Bad.*—If the immediate roof consists of a foot of draw slate which has no beam strength and tends to fall when the coal is blasted, the jacks are arranged as shown in Fig. 3. The single outer row of jacks is called the break row and is spaced on 3-ft. centers. The next row of jacks is called the draw-slate row, is spaced on 3-ft. centers, and supports the outer ends of steel beams which serve as forepoles. This beam is an "H" section, is 15 ft. long, and is placed in a round hole drilled just below the roof 7 ft. deep into the solid coal before undercutting. The third row of jacks is called the safety row and consists of light pipe jacks to prevent any tendency of the steel beams to sag. A complete

cycle of operation, starting with the conditions shown in Fig. 3, would be to blast the coal; to load it into the conveyor; to drill the holes into the new face to accommodate the forepoles; to advance the safety jacks; to advance the conveyor; to collapse the jacks of the draw-slate row seriatim, resetting them in a new draw-slate row; to collapse the jacks of the break row seriatim, resetting them in a new break row, and to undercut the face.

All three arrangements have been used in one or more of the nine longwall faces operated by us to date.

2. Depth of Cover

The depth of cover overlying the nine longwall faces has varied from 90 to 650 ft. To date there has been made no accurate measurement of the working pressures on the faces under these different depths of cover. We have, however, been able to make something of a check on relative pressures. In large testing machines at two universities we have observed the indentations made by the jacks in wood sills and collars under known loads, and we have compared these indentations with those obtained in practice in the mines. These observations lead to the tentative belief that, for a given thickness of coal bed, the working pressure on a longwall face will be approximately the same for shallow as for deeper mining.

3. The First or Major Break

The first few cuts on a new longwall face are made without the appearance of any pressure whatsoever on the break row of jacks or on the solid coal rib left behind. As additional cuts are made, and as the roof between the break row of jacks and the old coal rib is completely unsupported by any artificial means, pressure begins to show on the break row. This pressure continues to build up as the face sweeps forward until the point is reached where the strength of the immediate roof is not sufficient to support the beam between the break row and the old coal rib. At this point there is a more or less sudden collapse of the roof back of the break row of jacks, and this phenomenon is called the major break.

Just prior to this instant of roof collapse, the break row of jacks will have been called upon to carry its greatest load, and immediately following this collapse the pressure on the jacks will be almost nothing. In the case of the start of one of our nine faces, it was noted as the day shift left the mine that a major break was imminent, because the pressure was driving the jacks 1 to 2 in. into the wood collars and sills. When the evening shift entered the mine 2 hr. later, the roof was down beyond the break row, and the pressure was so relieved that a piece of cardboard

could in several cases be passed between the break row collar and the roof.

The distance which each of the nine longwall faces had advanced before the major break occurred varied between 70 and 90 ft. In each case we had planned, if 100 ft. had been reached without a major break, to drill jackhammer holes into the roof just behind the break row of jacks and to assist caving by the use of explosives. This, however, has not as yet been found necessary.

4. *The Daily Break*

We expect and almost invariably obtain a complete break throughout the entire length of the face each time the break row of jacks is advanced. Under a good quality shale roof this daily break may be counted on absolutely. Under a very strong sandrock, the break may not be complete each day at all points along the face. This however, has not proved serious because the jacks have shown the strength to carry the weight until the next succeeding advance completed the break.

The success which the jacks have shown in inducing a caving of the roof, and in thus relieving the roof pressures on the face, may be partially explained by the analogy of the straight line scratch of a diamond on a pane of glass. Just as the diamond scratch focuses the brittleness of glass along a line on which it is desired to break the pane, so the straight row of rigid jacks focuses the brittleness of the roof on the line at which we desire to induce caving.

5. *Occasional Increase in Pressure on the Break Row*

After the major break has been experienced, the normal pressure which is anticipated as the load to be carried by the break row may be subject to slight increases at intervals of 200 to 300 ft. as the face advances. It is believed that these surges of pressure are occasioned by readjustments of the strata in the vicinity of the previously described point at an unknown distance above the bed where the 45° breakline changes direction. These pressure increases have never been as great as those preceding the major break, and they have never lasted for more than 2 or 3 days.

6. *Jack Statistics*

The total number of jack-placements to date on the nine longwall faces is over 600,000. Each jack-placement means the collapsing of a jack after service at a particular spot and the resetting of it nearer the face as the longwall sweeps forward.

The theoretical breaking load for these jacks is 550 tons, and the safe carrying load is 200 tons.

The total number of jacks damaged under pressure has been two. Both were minor cracks in the outer edge of the base plate and were occasioned by flaws in the casting. Neither break caused the jack to fail in carrying its load.

A few jacks have been lost in the gob; but in each case the loss was sustained when longwall work was just being initiated and when the workmen were unfamiliar with their manipulation. The loss of jacks may be considered as negligible.

7. *Blasting*

The problem of blasting the undercut coal on long faces is entirely different from shooting coal in rooms. There are three desiderata: (a) minimum fines; (b) lumps not too large for easy handling, and (c) the least possible amount of pick-work to convert the back of the shots into a laterally straight and vertically erect new face for the next day's cutting. To obtain these results in the fullest degree, tests must be made under the actual conditions which occur at each mine.

In our own experience, in the four mines where we have blasted coal on long faces, there is considerable variation in the use of explosives. There are two points which are common to all four mines; the depth of the holes is such as to reach a point directly over the back of the undercut. and the spacing of the holes is on 6-ft. centers.

8. *Wage Schedules*

The wage schedules vary at our four mines; some are union mines, and some are non-union mines. Let us assume that at any mine the basic underground rate for common labor is W : then the different classifications of longwall work will be paid on approximately the following schedule:

Loading, hand-shovelers.....	1.0 W
Loading, machine operators.....	1.4 W
Undercutting, cutters.....	1.2 W
Undercutting, scrapers.....	1.2 W
Drill and shoot miners.....	1.2 W
Drill and shoot helpers.....	1.2 W
Jack operators.....	1.1 W
Haulage, motormen.....	1.2 W
Haulage, brakemen.....	1.2 W
Foremen.....	1.6 W

It may occasion surprise to note in the above schedule that in the case of men who work in pairs the helper is paid the same wage as the cutter, the driller and shooter, and the motorman. This is done for three reasons: (a) the work is more continuous and monotonous than men have been accustomed to in the past, and they like to "spell" each

other; (b) it permits the leader to train the helper so that management soon has two potential leaders, and (c) in case of the absence of the leader, the helper is fully qualified to step in and perform the day's task with a green helper.

9. Costs

Statements showing costs per ton in cents at the mines where we have operated long faces, or showing savings over room-and-pillar of 50 c. per ton at one mine, 74 c. at another, etc., would be valueless without an accompanying full description of the mining conditions. Lacking space in this paper for such descriptions, the writer gives below some of the units of cost in terms of man-days.

(a) *Undercutting*.—Two men, a cutter and a scraper, will cut a 400-ft. face $6\frac{1}{2}$ ft. in depth under average cutting conditions in 8 hr. The yield in marketable tonnage may be conservatively figured at 100 net tons per foot of bed thickness.

(b) *Drilling and Shooting*.—Two men, if equipped with an efficient boring machine, will drill, load, tamp and fire the 66 holes on 6-ft. centers along the 400-ft. face in 8 hours.

(c) *Loading*.—Hand shovelers will average 21 to 24 net tons per man, and will also advance the safety jacks and the conveyor under good roof, in 8 hours.

(d) *Advancing Break Row Jacks*.—A crew of 6 men will install the new double break row of jacks, will collapse the old double break row, and will build one cog along the loading heading in one shift of 8 hours.

(e) *Haulagemen*.—Two men will operate the conveyor, load the mine cars, and operate trains between the longwall face and the gathering parting in 8 hours.

(f) *Summary*.—It will be noted that 12 men on the face (cutters, drillers and shooters, jackmen, and haulagemen) are performing work which is independent of the thickness of the bed of coal. The number of hand-shovelers is dependent on the tonnage. The following table shows the resulting tons per face employee per day for varying bed thicknesses, exclusive of development work, maintenance of equipment, and inspection, on a hand-shoveling basis:

Thickness of Bed, Ft.	Tons per Day on 400 Ft. Face	Number of Shovelers	Other Laborers	Total Employees	Tons per Total Employee
3	300	14	12	26	11.5
4	400	19	12	31	12.9
5	500	23	12	35	14.3
6	600	27	12	39	15.4

Scrutiny of the above table shows the tremendous advantage which will result from the use of a loading machine in thicker beds. Two

shovel operators and four helpers to do the work of cleaning up, moving the safety jacks and advancing the conveyor, can replace all of the hand-shovelers shown in that table. Our own experience has not covered the use of mechanical loaders because our beds have been too thin for the loaders now on the market. However, we believe our method is perfectly adapted to loading machine operation, and we submit the following table to show the expected results from complete face mechanization under good roof conditions. This table assumes that the capacity of the loading machine will vary with the thickness of the coal bed. The table is exclusive of development work, maintenance of equipment, and supervision.

Thickness of Bed, Ft.	Tons per Day on 400 Ft. Face	Total Employees	Tons per Employee
3	300	18	16.7
4	400	18	22.2
5	500	18	27.8
6	600	18	33.3

CONCLUSIONS

In any attempt to express in a single paper the results of four years of study, investigation and experimentation, there must of necessity be a severe limiting of the phases to be discussed. Some readers of the above paper may be disappointed at the omission of such subjects as the following:

- A. Application of longwall to thick beds of coal.
- B. Application of longwall to steeply pitching beds.
- C. Development layouts for longwall under various mine conditions.
- D. Details of organization to accomplish the daily cycles of operation.
- E. Labor's reaction to complete face mechanization as evidenced by studies of absenteeism and turnover.

F. Discussion of the types of various pieces of face machinery; such as, conveyors, undercutters, loaders, drills, jacks, etc.

G. Discussion of the proper maintenance of face machinery.

H. Proper methods of supplying power on long faces.

These subjects are of such vital importance in longwall that each should be made the topic of a later paper before the Institute. However, lack of space and in some instances lack of sufficiently complete actual experience ruled them out of the present consideration.

Readers of this paper will have noted that its scope is confined to the fundamentals of the problem of complete face mechanization, these fundamentals being substantiated by a brief summary of the results obtained through actual experience.

It seems very opportune that this preliminary paper on longwall should have been prepared for presentation at a meeting where mechanical and mining engineers are in joint conference. Heretofore the mechanical engineers have designed and placed on the market what looked like very dependable machinery and have urged that it be given a chance to operate continuously. The mining engineers, however, have been far behind them and have failed to modernize the layout sufficiently to give the machines the opportunity to prove their efficiency. In our experimentation, we have found that some of the machines, which gave every indication of dependability, were unable to stand up when given an opportunity to operate continuously. The manufacturing concerns back of these machines have been ready and anxious to make the necessary corrections, but we are still a long way from perfection. A full measure of success in the future can be secured only if the mechanical and mining engineers work out the many details of complete face mechanization jointly.

DISCUSSION

R. Y. WILLIAMS.—I would like to make one or two remarks on the general situation. All other industries, steel, automobile, power, light, radio, have developed to the point of quantity production accomplished by complete mechanization, but in the coal industry the work of the miner at the face is done identically as it has been for the past years and years. There has been little change.

In the coal business, between 60 and 70 per cent. of the total cost of production must be paid to unsupervised labor because we have not quantity production through complete mechanization, and the result is that the public looks upon the coal industry as the most wasteful and inefficient of all the big industries—wasteful not merely in the loss of natural resources, but in that waste of human endeavor which today ought to be and could be so much more cheaply and efficiently done by mechanical power. The strange part about it is that mechanical engineers have placed on the market for the coal people dependable machinery for cleaning, for sizing, for dumping, for undercutting, and for drilling holes, and are today working out machines for loading and for conveying. Actually the only single operation in connection with the mining of coal that is without some mechanical thought is the question of timbering. And yet, despite all this wealth of machinery we are still paying that 60 to 70 per cent. of the total cost of production to unsupervised labor. What is the reason for it? What is the obstacle that stares us in the face? It seems to me, it is because we mining engineers have failed to develop a method of mining at the face that will permit machinery to operate continuously and economically. In other words, the obstacle is the absolute insistence on operating on the antiquated room-and-pillar system.

C. EVANS, Scranton, Pa.—This development of the longwall system is regarded as one of the most promising, although the most radical step, that has been made in mining in this country, or so far as our knowledge goes, in any other country. When Mr. Williams speaks of this paper as covering the requirements for the complete face mechanization in coal mining, I am afraid he gives the impression that it is something that can be applied pretty generally. It can be if the mining engineering industry can be persuaded to abandon its old methods, but I fear that there are hundreds of men in the mining industry who are not prepared to do this.

About 98 per cent. of the coal that is produced in this country today comes from room-and-pillar mining. Mr. Williams deals entirely with the advantages of longwall mining. Before we can get a complete mechanization, we must go through a development stage that includes mechanization in room and pillars. Mr. Williams' system is practical under many, many conditions, but under a great many other conditions—in old mines, particularly in the anthracite region—there is little or no possibility of introducing longwall. Nevertheless, it is necessary to study roof control in room-and-pillar work, and I think that Mr. Williams' paper will be a distinct contribution to those in the industry who will take the trouble to study it and try to appreciate the new points.

Our company does much work that we call longwall, but it is not true longwall because we have never attempted to properly control the roof. I believe that that is true of practically all of the so-called longwall that is done in this country, with the possible exception of some in Illinois.

For the complete mechanization of mines, mechanical engineers will have to do more work on points other than those connected with the moving and loading of the coal; they must devise more efficient and more dependable systems of transportation. In underground work it is difficult to get an adequate car supply to a point at which concentrated mining is taking place. The amount of track is so limited, or the room that is available for track is so small that some system must be worked out of greater capacity of mine cars, greater capacity of locomotives, better dispatching systems; so that the coal after it is moved out in the quantities Mr. Williams mentions—600 tons from one point of origin—can be taken away to the railroad cars.

Mr. Williams said, I believe, that there has been no change in the work of the miner at the face in many years. Unfortunately, that is too true; but I must say, in defense of the industry, that within the last five years there has been a very distinct effort to improve the quality of that work. This can only be done, of course, through supervision, and in the ordinary room-and-pillar mine supervision is a very difficult thing to get, to maintain and continuously apply, because a single foreman will have under his charge 20 to 40 separate working places, each one with insufficient light and each separated from the other by a solid pillar. All of these things must be taken into consideration by any one manufacturing mechanical equipment to be used in our mines as they are operated today.

The company that I am employed by has 5000 working places in all of its mines. Of those, 168 are longwall faces; modified, not ideal faces such as Mr. Williams is talking about—long faces varying from 185 to 270 ft. The face in the ordinary chamber is about 24 ft. We have been operating long faces in increasing numbers ever since 1916. We gradually increase the number each year as the territory capable of that sort of mining becomes available, and we would be delighted if underground conditions in the rest of our mines would permit us to change all of our faces to longwall. We are getting more coal per man, we are getting cheaper coal from the longwall faces than we are from any short face. It is true that the coal from our longwall faces is dirtier than that from the short faces, but that is because all of this coal is loaded by mechanical means, which practically prohibit handpicking of refuse as loaded, as is done in all hand-loaded places.

L. C. CREWE, Philadelphia, Pa.—I have had considerable experience in mining bituminous coal, particularly in Tennessee. It seems to me that generally speaking the points made by Mr. Williams are well taken. It stands to reason that the mines can be more easily mechanized by the use of the longwall system, and on at least two occasions I had a study made for my company looking to that end, but I ran up against the fact that the cost would be prohibitive. As I remember, there are something like 6000 bituminous coal mines in the country, and if 98 per cent. of the coal is now

mined by the old room-and-pillar system, and that system modified, including various forms of the panel system, it would appear that to change the chief production of the country from the old system to the longwall would be a Herculean task, not only from the management and labor standpoint but also from the standpoint of the enormous amount of additional capital that would be required. Assuming we all agree that, as a general proposition, the longwall system is the thing, how could we introduce it except by the expenditure of a prohibitive amount of capital, and gradually work to it by the training of management and of men over the period of a good many years? That is my query.

MEMBER.—Does this system of mining mean a return of 20 per cent. on the money? How does that stack up in savings?

R. Y. WILLIAMS.—We would require one undercutting machine for every five machines now used in a room-and-pillar mine to get the same tonnage. That leads to an actual saving. If you buy a loading machine for a room-and-pillar mine in order to save, you can buy one for a longwall mine and save five times as much.

Actually, it would cost about 20 c. per annual ton to equip a face 400 ft. long with jacks, with a longwall undercutting machine, and with a conveyor. In soft coal I imagine the cost is a little over \$3 per annual ton for plant and equipment. By the addition of 20 c. per annual ton you can save from 25 to 50 c. per ton of production.

R. D. HALL, New York, N. Y.—My preference is for longwall rather than for room-and-pillar workings. At the same time, I cannot help believing that Mr. Williams was incorrect in saying that the room-and-pillar system has failed because we have had that method of working for 100 years and have made no progress with it. That system of mining has a chance to progress that it never had before. Undercutting and loading machines and conveyors have been introduced; also better track systems. Something has been learned about management in connection with room-and-pillar workings. With those changes the room-and-pillar method is giving results it never gave in earlier days. Longwall success depends largely on these selfsame new types of equipment and management methods.

I was in a mine the other day where the coal was 42 in. thick, and the three men at the face, with one man receiving coal at the end of the conveyor, were loading together 90 to 107 tons daily from a single room. That is much more than was done in the past. Mr. Williams said that, with room and pillar, 20 rooms were needed for 100 tons of daily production. Doubtless that was true under the former conditions, but here was one room producing nearly 100 tons daily. The room-and-pillar system is costly to install with modern appliances. If we do not want to face the expense for modern equipment we can not get the best results from room and pillar or, for that matter, from longwall either.

Mr. Williams shows how much saving in ventilation cost is effected by the introduction of longwall. Unfortunately, if the fan is already installed and gives a large quantity of air, it will have to run inefficiently if a system is introduced that does not give an equal resistance to the air current. The entire advantage of longwall will not be enjoyed as soon as longwall is installed.

Much, it is true, may be said in favor of longwall, especially as to the larger sizes of coal obtainable thereby. For instance, in one mine at Paris, Ark., the coal is dropped from a longwall face on to a plate or "scow" and hauled to low cars in 1-ton chunks, no powder being used to bring it down. What I contend is that room and pillar can be made more economical than it is now and that Mr. Williams has shown it in his address at its earlier worst rather than at its present best.

Some of the most interesting features of Mr. Williams' paper are his references to roof action. We have been hearing for years from Europe and from our own country

that when a large area of mineral has been extracted the lower part of the roof would fall in large quantities in such disorder that it eventually would fill the open space and hold up the roof. That seemed ridiculous because the later increments became smaller and smaller and the rock could not fall till it choked. It cannot fall if it is supported and if it falls it drops where it can no longer support the roof from which it fell. Now we learn from Mr. Williams what really happens. The rock piles up from the bottom and the roof subsides so as to sit on it. On this pile of rock the roof ultimately rests after it has crushed it down to the required density. The roof *does* have to subside but not down to the floor whether heaved or natural, but on to the pile of detritus that its failure has created. Except with abnormally thick beds it need not fracture all the way from the extracted coal bed to the surface or from the surface to the extracted coal bed.

The main roof in most cases acts as a monolith, a single body, and gradually subsides toward the floor, fracturing in so doing. In the course of that subsidence it breaks on the surface back over the unexcavated coal. Because this break on the surface has been found it has been thought that it extends vertically or at some angle clear down to the coal bed. But there is nothing to show that this crack extends so far. Some in England, seeing the 45° break near the coal face extending over the excavated area, have said it reaches back to the crack on the surface, but there is no reason to believe the two fractures meet or are in any way one and the same fracture.

The correct theory is that the main roof fractures in the midspan of the excavated area and that then a lower layer of the roof begins to peel away from the main roof just as a layer of wood in a bent sapling will peel away from the main stem if it is cut or broken at any point. This layer which is thus freed is thin and so has no strength. It quite naturally breaks off at the rib on an inclined line and not at right angles, the fracture starting, of course, at the top away from the rib but sloping at an angle toward it. As a rule, it is said, the roof fractures so that it slopes upward and outward over the excavated area but the fracture actually starts in the other direction.

It is part of this lower layer that Mr. Williams holds up with his jacks and not the main roof. The latter, he would never be able to sustain. I have tried to visualize a jack that would hold up a roof 600 or 1000 ft. thick. Think what it would mean to work with jacks under the Woolworth building with the jacks resting on clay and the building not a mere shell like that or any other office building but solid masonry from top to bottom.

Evidently Mr. Williams' jacks hold up only the lower layer of roof, leaving the upper part to sustain itself by its own strength as long as it can. This lower layer is not a cantilever in reality because a cantilever presupposes an ability to "cant," or teeter. This beam cannot cant because it is held at one end as in a vise by the weight of the main roof pressing on it and thus on the coal below. It is a "built-in" beam or beam *encastré*. This beam bends down near the rib under its own weight. It separates vertically from the main roof unless the jacks hold it up in place but if they do suffer it to sag a little it is likely to spring up when a large part of it breaks off and thus lightens the load on the part left. Thus, where the jacks have been subjected to tremendous pressure the lower roof slab may lift enough so that boards can be placed between the slab and the jack or the blocks above it. Mr. Williams in his paper describes such a movement which the remarks I have made, in my estimation, fully explain.

Mr. Williams' excellent paper is subject only to the criticism that it does not give expression to the possibility of improvement in the working of coal by room-and-pillar methods.

T. M. CHANCE, Philadelphia, Pa.—In ordinary methods of mining, face preparation is considered to be of paramount importance. In many ways this is an unfortu-

nate situation because every time the miner removes so-called waste from the coal he is required to determine, under the poorest of lighting facilities, whether the material is fit to be loaded or should be stowed as waste. He thus becomes an inspector charged with the determination of the final disposition of property belonging to the owner of the coal. This is hardly fair either to the miner or the coal owner.

On the other hand, every system looking towards increased mechanization underground tends to reduce the possibility of such face preparation and therefore throws the burden of cleaning to the surface where it properly belongs. I do not mean to say that waste material that is pure rock and of sufficient size to be economically gobbled should not be left in the mine, but I do believe that a fair percentage of the present work now performed at excessive cost inside the mine will inevitably be transferred to the outside wherever mechanization is extensively practiced.

In regard to the particular problems that Mr. Williams has brought before us, I would call attention to the necessity of considering the local conditions that may be present. We frequently think only of the roof pressure in terms of vertical depth below the shaft collar or slope entrance but at times the increased rock pressure produced by hills rising high above the mine entrance may be of great importance. Thus at Coal Creek, Tennessee, the rock pressures that must be overcome as the coals are mined to the dip become excessive, due to the surface topography, and these seams must be won under conditions present in deep shaft mines in level territory.

Longwall in its earlier forms has been extensively used in both the Illinois and northern anthracite district of Pennsylvania. In 1915 the Delaware, Lackawanna & Western Co. had installed a number of face conveyors and cutting machines and had thoroughly developed a method of longwall involving the use of timber cribs. At this time Cadwallader Evans initiated mechanization in the room-and-pillar workings of the northern anthracite field by the introduction of the scraper loader and this method of mechanically working room-and-pillar operations has been extensively developed in many parts of the anthracite field, the Colonial Colliery having a number of miles of haulage rope in service for these scrapers.

The two forms of mining have therefore been mechanically developed to a fairly high degree but without introducing mechanical methods of controlling the action of the roof other than by attempts to do so by the use of timber, permanent pack walls or filling. I think Mr. Williams has presented something that is basically new and that should not be confused with longwall mining as we have known it in the past. There can be no question that an orderly system of controlling the whole mining operation will result in the same increased efficiency effected by modern methods of routing material through manufacturing plants. Such a systematized method of mining is impossible if we are to depend on natural forces for the control of the roof and maintenance of the working places, because such natural forces vary widely on account of the existence of faults, variations in the structural strength of the rock, and many other conditions, local or general.

T. F. DOWNING, JR., Philadelphia, Pa.—I do not like to hear people try to leave the impression that the coal business has not advanced. I have read that so often, and it is not a fact. The last figures available from the government were in 1924, and they show that the bituminous industry has increased its efficiency per man employed so that in 1924 only 170 days were required to produce what would have required 268 days 20 years before. That is one answer to the overmanning of the industry. The industry is growing, advancing beyond the capacity of the market and the men to produce.

I have seen much longwall in Europe; I went over there to study it and came back and tried it, and while I admit I did not use the jacks, my cost was 19 to 25 c. greater

in that section than it was in any other section of the mine, and a man was killed in trying it.

Of course, we could do away with it. It was not the trouble of the system so much as I think it was in the roof. There are many places in this country where longwall can be used to great advantage; but there are many other places in the country, particularly where there are special-purpose coals, where it cannot be used. Unfortunately, the best coals seem to have the worst roof. Gas coals as a rule have worse roofs than any others.

In this discussion the room-and-pillar system has not been given the best of the argument. There is one thing that we must consider; that is that only from 40 to 50 per cent. of the coal is taken on the advance. If the miner or the owner leaves the 50 or 60 per cent. to be taken out 5 or 10 years from now, he is certainly going to have a lot of trouble doing it. If the room-and-pillar system is used to its best advantage, the result is practically what is obtained with the longwall face. That is, if the rooms are driven no sooner than the pillar on the side of the room can be pulled back, there is practically a face that can be run as high as 1500 ft. long without any trouble; and in that way there is a grouping together of men, a concentration which is necessary for economic operation.

The feature of time has not been touched on in this longwall system. With the room-and-pillar system, I think mining men will agree, the faces can be protected for a greater length of time than on the longwall, so time enters into the discussion to a great extent.

Mr. Williams talked about working longwall for two shifts. As a matter of fact, the mines of this country do not work their places half a shift. That seems to be a pretty strong assertion, but actually the mines of this country produce less than 40 per cent. of the amount of coal that they are equipped to produce. It may be that there are some mines that work six days a week, and others that work two, but the average is less than 40 per cent. actual production of what they are equipped to produce. If we are going to have this longwall face to protect half of the time that the mine is not working, it is going to add to the cost of that system much more than a group of narrow openings in the coal. That is a feature well worth considering before going to longwall; that is, will my markets allow me to work every day in the week, or practically every day in the week so that I can get this 400-ft. line of jacks ahead every day, or must I leave them there for several days until the roof is cracked around them and broken into the face? If you lose your longwall face, which you say is producing 600 tons a day, you have cut down your production 600 tons until you get a new face, whereas if you lose the face on a narrow pillar, you have lost very little. In other words, 99 times out of 100 you can place the man in another pillar if he is willing to go, so that system does not lose you the tonnage.

R. Y. WILLIAMS.—That is so important, may I say one more word. With jacks the work is different from that with timbering. About a year ago, we had occasion at one mine to shut down three long faces for four months. At the end of four months the order went in to start the three faces. Two of those faces were started without one iota of work required to clean them up. The third face required about six men for 8 hr., and the reason for that was that that face happened to be in a fault at the time that the order to shut down took place, and at that point there was a heavy fall.

The strength, the rigidity, and the movability of jacks will make you absolutely free of the union committee coming in and saying, "If you don't give us what we want, we will strike tomorrow and you will lose your face." We have never had any suspicions of anything causing us to lose the face. We have passed, under faults, and rock cracks almost parallel with the face—one was big enough to get one's arm in—

and yet we swept right under. That four months' stoppage was one of the most interesting lessons we learned from our control of roof on longwall faces.

J. B. DILWORTH, Philadelphia, Pa.—I take it from Mr. Williams' paper that his longwall system of mining is predicated largely on the successful use of loading machines. It seems to me that this is its main weakness; though certainly if these machines can be used successfully at all, their highest efficiency should be shown when working along such longwall faces. The main positive results of these experiments then, as Mr. Williams has so well brought out, is that the jacks have solved the problem that has almost universally stumped the longwall miners in this country—that is, the problem of holding up the roof. But even though these jacks solve the big problem of roof control, a difficulty that still remains is the effective loading of coal by machines. A loading machine will not work very well in thin coal, and if there are partings in the seam that means trouble when the coal is loaded out. Those two points, it seems to me, are very important factors because so much of the country's coal comes from thin or parted seams.

R. M. GATES, New York, N. Y.—There are more problems connected with this subject than the mere designing of various mechanisms. The mechanical engineers can, I believe, design machines to move anything or to support anything provided the mining engineers give them sufficient space for the machine to operate.

The capacity of a steam shovel is not the size of the dipper nor how fast it can move. Its capacity is limited by your ability to take the material away from the shovel or your ability to keep the shovel up against the face of the bank which you are working. This is just as applicable in a coal mine as it is in purely stripping operations.

The problem before us is largely one of cooperation. The mechanical engineer is not able to develop, in a satisfactory manner, a machine to do the many tasks discussed this evening unless he has complete knowledge as to the factors that will enter into the economic operation of such a machine. It is not always possible for the inventor or designer of such class of equipment to be experienced mining operators. Therefore, it is essential that there be very close cooperation between the mining engineer and the mechanical designer.

Generally, in the successful development of a new device, it is necessary also that the operator change his methods as to give an opportunity for the best functioning of a machine. In my experience in the mining field, I have felt the great lack of this cooperation which is essential for the successful development of machinery which you operating men can use effectively. The success of a particular machine is dependent upon the proper functioning of the other machines with which it is in series; also those factors that directly or indirectly affect its capacity.

H. LOUIS, Newcastle-on-Tyne, England (written discussion).—While I can fully believe the advantages claimed for mechanized longwall working in the cases quoted by Mr. Williams, and am prepared to agree that similar advantages can be obtained in a number of similar cases, I feel compelled to enter a caveat against the dogmatism implied in his paper that such a method must necessarily be best in every case. I have known mines where room-and-pillar working was replaced by longwall working with much advantage, and I know others where room-and-pillar working is in successful use today and where longwall would be hopelessly impracticable. To quote a sentence from H. F. Bulman's recent book, "The Working of Coal and Other Stratified Minerals" (p. 83): "Generalizations are generally misleading." There is such an infinite diversity in the mode of occurrence of coal seams that the only safe statement to make is that there is no one best method of working all of them but that each case requires separate consideration. The comparison instituted by Mr. Williams between

his mechanized longwall and what he is pleased to call room-and-pillar working breaks down on account of the fact that his so-called room-and-pillar is not room-and-pillar working at all as we understand it in this country. No better evidence of this could be given than is found in the phrase on page 239 of his paper: "If pillars are later robbed." This "if" gives his case away; we got beyond that over 100 years ago, and no one would dream today of laying out room-and-pillar workings save with the intention of keeping his pillars of such a size that they can safely be won off. In this respect, it is a matter of indifference whether the narrow work is completed first before "broken working" commences or whether the latter closely follows the narrow work. What Mr. Williams calls room-and-pillar working appears in fact to be a kind of stall working. If he wants to know what room-and-pillar working properly worked is like, I would advise him to read the work by Mr. Bulman, in which is given the definition: "In this method of working there are two stages: (1) the driving of headings at right angles to each other so as to form pillars. This is known as 'Whole' working . . . (2) the removal of pillars, which is known as 'Broken' working." Mr. Williams would then see the defects of his so-called room-and-pillar method. For example, he says that the number of places per 100 tons output per day averages around 20 working places. In one example quoted by Mr. Bulman (p. 93) he gives an example of a complete panel "containing 10 'whole' places, and 3 or 4 'broken' lifts." The output reaches 200 tons with an output per person employed per shift of 42 cwt., which is probably considerably less than Mr. Williams is assuming. In other words, modern British room-and-pillar is more than twice as efficient per working place as the method to which Mr. Williams applies that term. I hold that the mechanization of the coal face presents such obvious and such important advantages that there is no need to attempt to strengthen the case by unfair comparisons, but it must also not be forgotten that true modern room-and-pillar working lends itself quite well to a considerable amount of mechanization.

I take it that the method advocated by Mr. Williams is a sound one and would prove successful in a good many cases; I should object, however, to his method of leaving a narrow rib of coal on either side of his roadways instead of taking out the whole of the coal and replacing it by substantial pack-walls, as is far the more usual method. There may no doubt be cases in which the leaving of a rib of coal may be found to be advantageous, but in a coal liable to spontaneous combustion I can conceive of no means better adapted to invite that disaster than to leave a narrow rib of coal between goaf on the one side and a roadway on the other. Everyone who has had to do with coals of that character knows that it is imperative to get out every particle of coal as far as humanly possible, and above all to leave no ribs exposed to heavy crush and to air currents at the same time.

W. R. CHEDSEY, State College, Pa. (written discussion).—I have long been an advocate of the longwall system of mining for flat and gently dipping beds or anywhere else that it could be properly applied; also, I thoroughly believe in the ultimate economy of the retreating system, even though more capital is necessary in the early stages of an operation. This can be to some extent reduced by production from one area of a mine on an advancing face while development work is being carried to the proper boundary to start a retreating face or series of faces. I have had a little experience in longwall mining and have seen a little more on short trips to such mines and it has been my observation that on a full retreating system control of the roof pressure is more easily gained than on an advancing face where pack walls, entry pillars, or other supports must be left in.

Mr. Williams speaks of steel "jacks." I infer that they are not adjustable, at least through very much of a range of length, and I believe that they fall more properly into a class which years ago was called collapsible steel supports or props.

A point which Mr. Williams failed to mention is that occasionally the roof layers bend from the outer row of supports to the floor in the "caved" portions without any decided break. It was something of a surprise to me to find that rocks could be flexible enough to do this without more breakage.

The requirements of the perfect longwall face, as set forth in the paper, are undoubtedly an improvement over the old idea of longwall as being one continuous face around the entire mine, in that the work can be more systematized on the shorter or continuous faces as outlined by Mr. Williams.

Mr. Williams mentioned that his experience has been in beds too thin for loaders now on the market and I would be interested in knowing if they had tried, and with what success, the scraper loader instead of a shoveling machine and the face conveyor. I have seen scraper loaders used in several of the systems or so-called concentrated mining with complete success so far as the operation of the scraper is concerned except, perhaps, from the standpoint of degradation of some of the coal, and perhaps also of introducing a little more floor dirt into the coal as hauled from the mine.

M. J. CONWAY, Coatesville, Pa. (written discussion).—Mr. Williams has treated his subject with extreme clarity within the scope of his paper. Longwall mining if properly engineered is by far the most desirable method of coal getting in beds of medium thickness, but as pointed out by Mr. Williams, the various moves have to be carried out with a precision not usually found in mining methods and a mine that is planning on longwall development should also plan on a scheme of vocational training for the men who are going to be the key men in handling the system.

The importance of training as a precautionary measure can be readily seen. If something happens to prevent the getting of one longwall face for one day, the mine output suffers a loss of from 300 to 500 tons with greater difficulties to be experienced in obtaining the required tonnage on the next day.

Even with the pack-wall and brushing method of advancing longwall, tonnage can be produced in 3 ft. 0 in. to 5 ft. 0 in. seams on a cost basis comparable with room-and-pillar development with greater ease of supervision, ventilation and traffic control in addition to a certain degree of prevention of coal dust explosion due to the rock dust formed by the preliminary and back brushing.

In any type of longwall, too much stress cannot be given to the importance of keeping the wall straight, and the face should be frequently checked and "squaring up" cuts made as often as necessary.

In general, longwall mining lends itself to mechanization and therefore lower cost more than any other form of mining but a full measure of success will not be realized until the coal companies working this method train their men in the necessary fundamentals and teach them the reason for the sequence of the steps in the operation.

Blasting Coal Effectively and Safely in Southern Illinois*

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(New York Meeting, February, 1928)

FOR blasting in coal mines the U. S. Bureau of Mines recommends that permissible explosives be used exclusively, that these shall be fired electrically, and that where feasible the working place shall be cut or sheared. These recommendations are made to promote safety and prevent mine explosions.

Although permissible explosives are not flameless, their lower flame temperature and shorter duration of flame make their use safer than that of nonpermissible explosives. Mixtures of air with both methane and bituminous coal dust, which are liable to occur in coal mines, may be ignited by coming in contact with any type of flame or spark of sufficient temperature and duration or by the firing of a blast. In a working place where a blast is to be fired, in addition to the use of permissible explosives, adequate ventilation for the dilution and removal of accumulations or outbursts of flammable gas should be provided. Bituminous coal mines should be rock-dusted as a means of rendering the coal dust harmless.

The efficient use of permissible explosives depends upon the proper preparation of the working place, standardized drilling, the strength characteristics of the explosive, and the use of a proper weight of charge. Experienced shotfirers should load, tamp and fire all holes.

The major part of the information and data here presented is based on the results of efficiency tests made in large producing mines of Pennsylvania and Illinois. Bureau of Mines *Report of Investigations*, Serial No. 2697, describes how a standard method of blasting was worked out and put into operation in a mine in southern Illinois.¹ This paper contains a description of the methods worked out subsequently in another mine in that district.

Descriptions of tests in two Pennsylvania mines are given in *Bulletins* 11 and 19 of the Carnegie Institute of Technology.² These tests

* Published by permission of the Director, U. S. Bureau of Mines.

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¹ J. E. Tiffany and J. J. McKitterick: *Methods of Increasing Lump Coal Production, with Especial Reference to Southern Illinois*. U. S. Bur. Mines *Rept. of Investigations* No. 2697 (1925).

² J. E. Tiffany and C. W. Nelson: *Efficiency in Blasting Coal, Production of Lump Coal*. Carnegie Inst. of Tech. *Bull.* 11 (1924).

J. E. Tiffany and B. L. Lubelsky: *Factors Affecting Production of Lump Coal*. Carnegie Inst. of Tech. *Bull.* 19 (1925).

included quantitative determinations of the percentages of lump coal produced by two cuts in each of four adjacent rooms. The methods of blasting were varied, as were the brands of permissible explosives used. The natural conditions surrounding the tests remained as nearly similar as possible. In the two Pennsylvania mines, where tests were made, the Pittsburgh coal was worked and in the two Illinois mines the No. 6 bed of the Illinois series was worked.

SAFETY AND OTHER CHARACTERISTICS OF PERMISSIBLE EXPLOSIVES

The principal tests of permissible explosives made by the Bureau of Mines established their safety when fired in the presence of gas and finely ground bituminous coal dust. Other tests are made to measure the strength characteristics of these explosives, such as the average weight of a $1\frac{1}{4}$ by 8-in. cartridge; the unit deflective charge; and the rate of detonation. These characteristics have an influence on the effectiveness of the work done when these explosives are fired in blasting coal and govern the weight of charge necessary to bring down the coal.

Unit Deflective Charge

The unit deflective charge is that weight of an explosive in grams that gives the same swing on the ballistic pendulum as 227 g. ($\frac{1}{2}$ lb.) of Pittsburgh testing station standard 40 per cent. straight nitroglycerin dynamite; it is a measure of the propulsive strength of the explosive on a weight basis. In changing from one explosive to another, the proper charges to be used can be computed by knowing for each explosive the unit deflective charge and the weight of a $1\frac{1}{4}$ by 8-in. cartridge. A few examples will show the method of computation.

Computing Charges

First, assume that an explosive, which can be designated as brand "A," is being used in blasting the coal at a certain mine. Assume further that this explosive has a unit deflective charge of 235 g. and that each $1\frac{1}{4}$ by 8-in. cartridge weighs 170 g. Where three cartridges are used in a hole, the weight of charge would be 510 g. The use of an explosive "B" is contemplated in place of "A" because the former has a lower density, which usually tends toward a higher production of lump coal. The explosive "B" has a unit deflective charge of 245 g. and the weight of a $1\frac{1}{4}$ by 8-in. cartridge is 125 g. A greater weight of "B" is required than of "A" because their deflective strengths are as 235:245. It will therefore be necessary to use:

$$245 \times 510 \div 235 = 532 \text{ g. of "B"}$$

or

$$532 \div 125 = 4\frac{1}{4} \text{ cartridges}$$

A charge of $4\frac{1}{4}$ cartridges of "B" can then be tried to ascertain whether it will bring down the coal in a proper manner. Generally, a

change in explosive to one of less density will demonstrate either that the new explosive is not of sufficient strength to shoot down the coal properly and therefore cannot be used, or that the coal is brought down and shattered at the front of the cut. Subsequent charges should be regulated according to the results obtained in the trial shots.

Where a denser explosive "C" is proposed for use, the charge can be computed in a similar manner. The explosive "C" has a unit deflactive charge and weight of a $1\frac{1}{4}$ by 8-in. cartridge of 214 and 185 g., respectively. By computation, the charge is $2\frac{1}{2}$ cartridges. In this case it will be found that if the coal is not brought down at the front of the cut, the use of more explosive will probably cause undue shattering of the coal and therefore may not under such conditions be a satisfactory explosive to use in this particular instance.

Rate of Detonation

The foregoing considerations do not take into account the relative shattering effect of the explosive. To a large extent this effect depends upon the rate of detonation, which is determined by the Bureau of Mines by means of the Mettegang recorder.

In order to determine the effect of using explosives with varying rates of detonation, quantitative tests were carried out in a Pennsylvania mine where the Pittsburgh bed was worked. In these tests trials were made with five different permissible explosives. Four rooms were selected and cut twice for each brand of explosive. After the coal had been shot down it was loaded into the mine cars in the customary manner and hauled to the tippie. There, it was weighed as run-of-mine and then screened. The $1\frac{1}{4}$ -in. lump passed to a weigh-pan. The results were based on the mining of about 250 tons of coal for each type of explosive. There was 4.7 per cent. more lump coal produced when the result of blasting with an explosive, which had an intermediate rate, was compared with the one which had the lowest rate of detonation. Although these particular tests did not extend to other mines, the writers are of the opinion that there is one rate of detonation which will produce a maximum quantity of lump coal for any particular coal mine. Under a given condition however, this critical rate of detonation may vary with the nature of coal.

DIRECTION AND WIDTH OF WORKING PLACE

In many mines in southern Illinois which work the No. 6 bed, the approximate angle between the direction of the working places and the face cleat planes is 45° , because the working places are usually driven either north and south or east and west. The direction of the face cleats is usually about N. 45° W. In such places the working faces tend

to overhang, particularly if the holes are not drilled deep enough. It would not, in most cases, be feasible to change the direction of the working places because the layouts are usually too well established. It is therefore necessary to drill to the best advantage under the conditions presented. With possibly some exceptions, it is generally found that the highest percentage of lump coal is attained when the principal working places are driven on the face—that is to say, at right angles to the direction of the face cleat.

In a Pennsylvania mine additional tests, similar to those already described for the same mine, were made to ascertain the effects of driving on both the face and butt. The drilling and blasting were done in a uniform manner. It was found that in a butt entry 10 ft. wide and with the coal slightly more than 7 ft. thick, 54 per cent. of the coal screened on the tippie was $1\frac{1}{4}$ -in. lump coal. Trials were then made in places 18 ft. wide driven on the butt. Under this changed condition the lump coal percentage was increased to 56.8 per cent. On the other hand, when the working place was only 10 ft. wide and driven on the face, a 57.7 per cent. yield of $1\frac{1}{4}$ -in. lump coal was obtained. In rooms 18 ft. wide driven on the face the percentage of lump coal was 59.1 per cent.

USE OF FLAT HOLES

The effect of another variable was determined by making similar tests in which a comparison could be made of the use of holes drilled nearly flat—that is, almost parallel to the bedding planes—and those having a steep inclination. There was 5.1 per cent. more lump coal by the former method, which produced 64.2 per cent. of $1\frac{1}{4}$ -in. lump coal.

Suggested Procedure in Improving Blasting Practice

In studying blasting practices in mines and in devising improved methods, there are many things to be taken into consideration, such as the thickness and pitch of the coal bed; thickness of its cover; the number, thickness, and extent of partings; the definiteness, number and distance apart of the face and butt cleats, and toughness and brittleness of the coal. Current blasting methods should be studied. In southern Illinois, where the No. 6 bed is worked, 14 to 24 in. of coal is left in place for roof support. This roof coal is immediately above a persistent parting of mother coal which makes a plane of separation between the roof coal and the coal that is blasted. When a bed does not have a parting at the underside of the roof coal, the use of top-cutting machines can be considered.

PREPARING THE FACE

Aside from the fine coal arising from the firing of excessively large charges, the percentages of the smaller sizes of coal tend to increase with

the amount of pick work, which is required to bring down the coal from the face after the blast. The minimum amount of such pick work can only be realized by the proper carrying out of the operations involved in preparing the face. These operations will now be enumerated.

Cutting

The purpose of making a cut (under cut, middle cut, top cut, or shear) is to provide an additional free face, because two or more free faces in a working place afford a better chance for the explosive to bring down the coal than those with only one free face and aids in loosening the coal so that the product after blasting tends to be blocky and firm.

An under cut is generally the most efficient form of cut. A middle cut or over cut may be used instead of an under cut when some particular physical conditions such as the presence of shale or bone bands, impurities near or at the top of the bed, or bad roof conditions exist.

Cutting Machines and Mechanical Loaders

Cuts of many kinds can be made mechanically by machines. There has been an increased use of shearing machines in conjunction with mechanical loaders. Where a shearing cut (or vertical cut) is made, there must be a proper combination of shearing, snubbing, and placement of holes in order to obtain the best results by the use of mechanical loaders, because the results will depend upon the manner in which the coal is shaken up. The loading device should be able to take up all the coal readily and it is important to have the ribs properly cleaned.

Cleaning and Loading Out Cuttings

Studies by the U. S. Bureau of Mines have shown it to be essential that these cuts be made of uniform depth and square at the corners, also that close attention should be given to the cleaning out of the cut, so that all coal dust and other broken coal is removed before shooting. A scraper or long-handled shovel is the best tool for this purpose. The coal thus removed should be loaded in mine cars and removed from the working place before blasting. This will reduce the dangers arising from the presence of coal dust.

Relation between Blasting and Coal Loading

The importance of loading out all coal loosened by a shot before firing the next shot is shown by the results of two tests. In the first test all of the coal brought down or loosened (about 30 tons) by the rib shot, which was fired first, was loaded out before firing the second hole. This method resulted in the production, by actual screen test at the tipple, of 59.1 per cent. of 1¼-in. lump coal from the entire cut. In the second

test the quantity of coal loaded out before firing the second shot was only 8 tons. This was the minimum quantity that had to be removed in order that the miner could shear in along the rib to the back of the cut on the side where the first shot was fired. By this latter method there was only 56.5 per cent. of $1\frac{1}{4}$ -in. lump coal produced from the entire cut, or 2.6 per cent. less than by the first method.

Snubbing

Snubbing increases the height of the under cut and provides a means for the front portion of the coal to roll forward when loosened by the blast. As a rule the hand method of snubbing is employed.

One practice is to remove, by hand, from along the entire length of the face, with a wedge or pick, a wedge-shaped portion of coal from above the under cut. As a rule the snubbing is about 18 in. high in front and extends back until it tapers out at about one-half the depth of the undercutting.

Sometimes the snubbing is done by firing light charges of explosives in short low holes. Mechanical methods of snubbing have been tried but have met with little or no success.

Placement of Holes

The number and the placement of holes are by far the most important considerations in successful blasting. Of course, practice varies widely and depends upon natural conditions, which in turn depend upon such physical characteristics of the coal as to texture, hardness, friability, presence of impurities, the relative ease of its separation from the roof or floor, the weight due to the thickness of cover, and the pitch and thickness of the bed.

The number of holes required for a working place vary from two to five or more. Only two holes are used in some mines working the Pittsburgh bed, one at each rib. Where the production of lump coal is one of the major considerations, the best results are obtained by using as few holes as possible, provided that an extra large charge is not used in any one hole. On the other hand, it has been observed in mines where very thick beds of coal are worked that a large proportion of the fine coal is due to an insufficient number of holes. Where the number of holes per round is increased, the total weight of explosive originally used should be distributed between the additional holes.

The holes should be drilled only after the working place has been cut, otherwise their placement may not bear the proper relation to the cut.

Some tests made in a mine working the Pittsburgh bed showed that each rib hole should be placed 12 in. from the rib and the same distance below the roof, also that the hole should be almost flat and parallel to the

line of sights. However, in the tougher coal beds of southern Illinois, as represented by the No. 6 bed of the series, it is found necessary to increase the distance between the rib and hole to as much as 2 or even 3 feet.

METHOD OF DRILLING AND BLASTING IN SOUTHERN ILLINOIS

During a recent investigation in a mine in southern Illinois, the most desirable placement of holes was determined for both rooms and entries for this particular mine. Where the coal was loaded by hand, the holes were drilled by hand using augers mounted on posts. Holes were electrically drilled in rooms where mechanical loaders were used. The figures accompanying this paper are only intended to illustrate methods which were successfully used in two southern Illinois mines. For hand

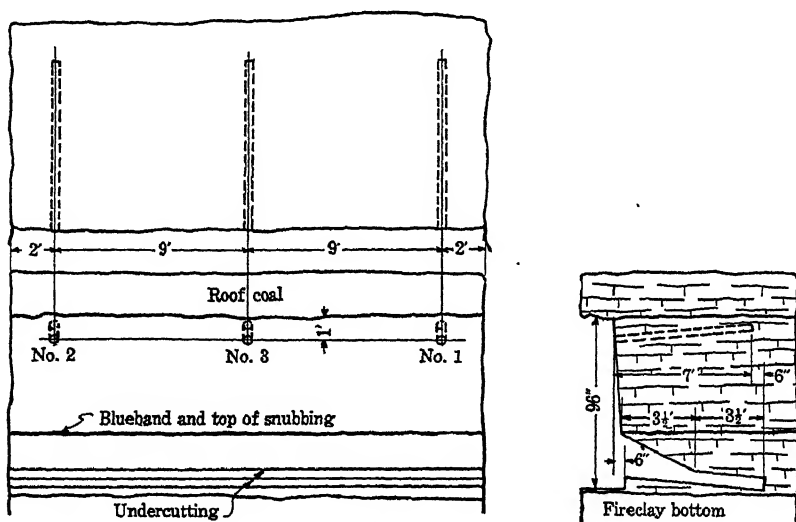


FIG. 1.—METHOD FOR DRILLING AND BLASTING FOR HAND LOADING IN ROOMS, No. 6 BED, SOUTHERN ILLINOIS.

Where it is feasible shots should be fired separately and the coal loaded up after each shot. Usually the center hole is fired first, then the rib holes. The charges of permissible explosive to be used for each rib hole should not exceed $1\frac{1}{2}$ lb. and for the buster shot in center hole $\frac{3}{4}$ lb. will usually be enough.

loading in rooms, the placement of holes is as shown in Fig. 1 and for entries as in Fig. 2. For mechanical loading in rooms, the methods shown in Figs. 3 and 4 were used. Figs. 1 to 4, inclusive, refer to methods employed in mines working the No. 6 bed of southern Illinois. As previously stated, in these mines 14 to 24 in. of coal is left in place to support the roof, and there is a persistent parting of mother coal on the underside of the roof coal. In the upper third of the bed there are several other partings of mother coal. The chief impurity in the bed is the "blue" band, which is a gray shale parting of 1 or 2 in. in thickness, about 30 in. above the bottom of the bed. Locally, the bed contains pyritic impuri-

ties in the form of lenses and boulders; the latter are found below the blue band. In Figs. 1 to 3, inclusive, the holes are placed 12 in. below

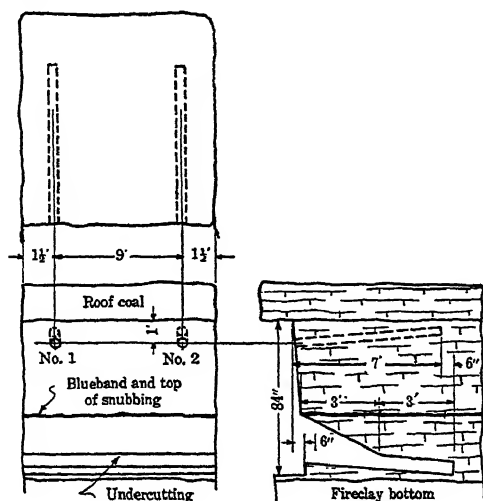


FIG. 2.—METHOD FOR DRILLING AND BLASTING IN ENTRIES, NO. 6 BED, SOUTHERN ILLINOIS.

When the coal is not snubbed an additional hole should be drilled in the center of the entry just above the blue band. The charge should be $\frac{3}{4}$ lb. of a permissible explosive. Not more than $1\frac{1}{2}$ lb. of a permissible explosive should be used for each hole and each hole should be fired separately.

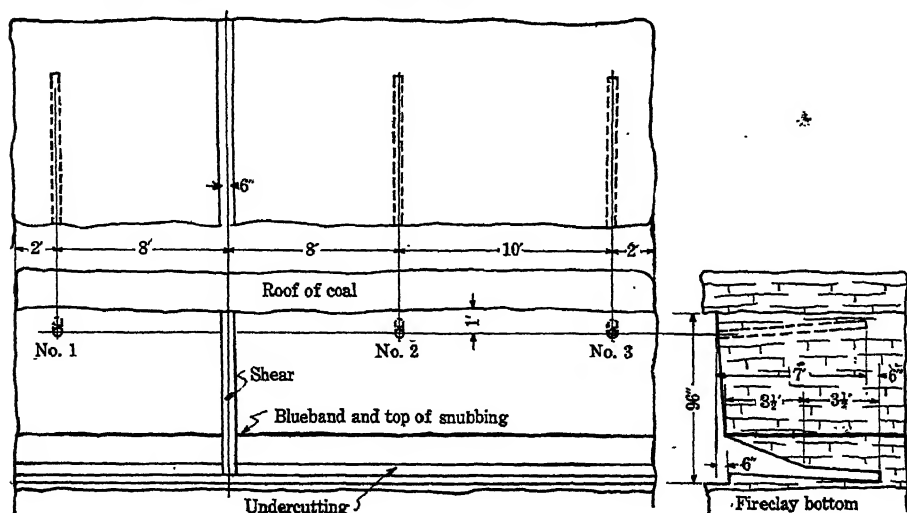


FIG. 3.—METHOD FOR DRILLING AND BLASTING FOR MECHANICAL LOADING IN ROOMS WITH ONE SHEAR, NO. 6 BED, SOUTHERN ILLINOIS.

Not more than $1\frac{1}{2}$ lb. of a permissible explosive should be used for each hole and each hole should be fired separately in the order as numbered.

the roof parting. They are nearly flat and do not rise more than 4 in. in their entire length.

Hazard in Drilling into the Roof Coal

There is a tendency to incline the holes to such an extent that they are frequently drilled through the parting and into the roof coal. This

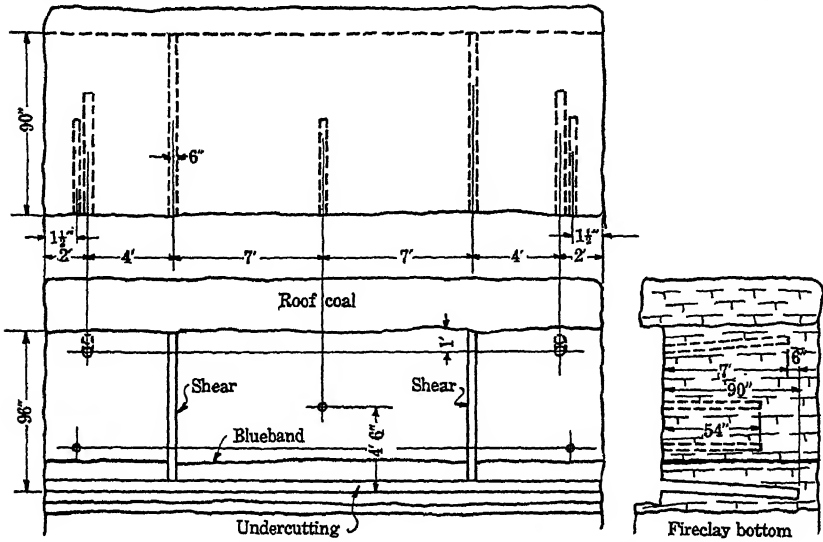


FIG. 4.—METHOD FOR DRILLING AND BLASTING FOR MECHANICAL LOADING IN ROOMS WITH TWO SHEARS, NO. 6 BED, SOUTHERN ILLINOIS.

The shots are fired separately, first the lower holes and then the upper holes.



FIG. 5.—FALL OF ROOF DUE TO IMPROPER SHOOTING.

practice is very bad and increases the dangers of mining. The effect of drilling into the roof coal of the No. 6 bed in southern Illinois is to

break it and allow the shale above the coal to fall, as shown in Fig. 5. The operations involved in cleaning up a fall and timbering the broken roof make additional costs and hazard.

In some sections of a Pennsylvania mine which was investigated, it was found that the coal would not separate readily from a slate roof. Here it was advisable to incline the holes so that they would just touch the roof at the back of the hole.

Direction of Rib Hole

It has been pointed out that a rib hole should be parallel to the line of sights. However, it is common practice to drill it in a direction that is parallel to the side of the undercut, and if this is not square such a hole is liable to be drilled on the solid. Usually the driller is not aware of this because he does not always take into account changes in the direction of the side of the undercut that usually occur near the back of the undercut, which has not been made square.

The firing of a hole drilled on the solid is very dangerous and gives poor results; it is liable to be the cause of a blown-out shot, and also of badly shattered coal adjacent to the hole.

Overhanging Faces and Ribs

Overhanging faces and ribs are caused principally by poorly placed or shallow holes. When holes are drilled only 6 ft. deep for a 7-ft. undercut

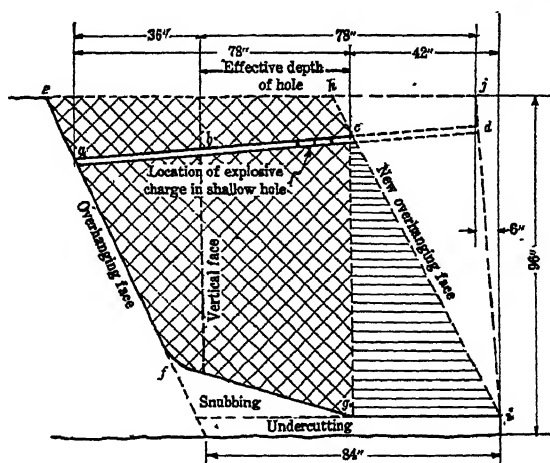


FIG. 6.—DIAGRAM SHOWING THE EFFECT OF SHALLOW HOLES.

there is a tendency for the overhang to increase about one foot after each cut has been cleaned up. Only that portion of the hole which is vertically above the undercut can be considered effective (see Fig. 6). In one southern Illinois mine 13 rooms were visited at random and measurements made of the three holes in each room. The average depth

of hole was 76 in. with an average overhang of 25 in. Fig. 6 shows a section in one of these rooms where there is an overhang of 36 in. One of the holes was drilled to a depth of 78 in., as shown. This would have been the proper depth of hole for one drilled in a vertical face from *b* to *d*, but with the overhanging face the depth of the hole should be increased 36 in. or the amount of the overhang at the mouth of the hole.

The firing of a charge of explosive in a shallow hole *abc* will tend to shatter that portion of the coal represented by *efgh*, while that in the portion *cgi* is broken only to a slight extent and can be brought down only by an excessive amount of pick work. The portion *hji* is not affected by the blast. The overhang at *h* has been increased by 6 in. For the most efficient blasting the quantity of coal in the portion *efgh* should be as large as possible and that from *cgi* as small as possible.

Depth of Hole

The hole should be drilled 6 in. short of the undercut. A vertical line projected from the back end of a hole which has been drilled to this depth will intersect the plane of the undercut at 6 in. in front of the back of the undercut.

Shorter holes than these cause the face to overhang. Faces overhanging from 3 or 4 ft. were frequently observed. The excuses made for gripping a cut at the ribs have been that an overhang of face or ribs existed that would not permit the proper operation of the undercutting machine.

Another claim somewhat frequently made in mines operating in the No. 6 bed of southern Illinois is that the explosive breaks the coal ahead of the end of the hole and toward the back of the cut, but this claim was not substantiated in any of a large number of observations in one of these mines, although it might be so under other conditions.

Charges to Be Used

After conditions have been improved by the proper preparation of the face, attention can then be directed to any necessary reduction in the weight of the charge of explosive used. The charge should not be reduced to the extent that the coal is not brought down with certainty.

Air-spaced Shots

The shattering effect of permissible explosives can be reduced by the use of air-spaced shots. There are a number of ways of doing this, but the one that generally gives the best results is by reducing the diameter of the cartridge of explosive, when the diameter of the drill-hole remains the same. In one of the investigations in southern Illinois the size of cartridge was changed from $1\frac{3}{4}$ by $6\frac{1}{2}$ in. to $1\frac{1}{4}$ by 8 in. By this and other changes the quantity of explosive used was reduced by 10 per cent., and

there was an increase of 2 per cent. in the quantity of 6-in. lump coal produced. However, in other mines where tests were carried out the weight of the charges could not always be reduced when air-spacing was used, and the advantages of air-spacing were confined to improvements in the firmness and size of lump produced.

Method of Charging Air-spaced Shots

Where the shots are air-spaced by reducing the diameter of the cartridge, all of the cartridges required for a charge should be placed in the mouth of the hole at one time because this procedure minimizes the lodgment of coal dust between the ends of the cartridges. The cartridges should not be slit and rammed. The cartridges should be pushed to the back of the hole with a wooden tamping stick.

Use of Stemming

Noncombustible stemming only should be used; coal dust or slack invites disaster by explosion. Clay is a very suitable material for stemming because it is plastic when moist and can then be made into "dummies." The first two dummies should be pressed lightly against the explosive. The tamping of the remaining portion of the stemming should be done with greater force until the borehole is filled to the collar. Before firing, the place should be inspected for gas. If methane is found the shots should not be fired until the methane has been removed.

Firing the Shot

A firing cable not less than 100 ft. long should be used. The shotfirer should use a permissible single-shot blasting unit. The shot should not be fired except from some point where the shotfirer will be protected from flying pieces of coal. He should always give warning to the other workers prior to firing.

RECOMMENDATIONS FOR PRODUCTION OF LUMP COAL

The following recommendations, if properly carried out, should increase the production of lump coal:

1. The principal working places should be driven at right angles to the face cleats.
2. In order to control the placement of holes it is preferable to have them drilled by shotfirers or by others experienced in drilling coal, rather than by the miners. The loading, tamping and firing of shots should be performed by shotfirers and not by miners.
3. Overhanging working faces should be eliminated as much as possible. These arise generally as a result of the holes being of insufficient depth or poorly placed, or both.

4. The use of water jets on the cutter-bars greatly reduces the production of and danger from dry coal dust. The undercut should be of uniform depth and square at the corners. A straight rib line should be maintained.

5. Before blasting, machine cuttings (bug dust) should be removed from the cut and should be moved several feet from the face, or preferably loaded out.

6. The exclusive use of permissible explosives, electrically fired, is recommended for coal mines and if feasible, the working places should be cut or sheared.

7. Before loading the holes with explosive and stemming, the coal dust, which was produced in drilling the holes, should be removed as completely as possible.

8. The charges for each type of hole should be standardized.

9. The holes should, wherever feasible, be fired separately and the coal loosened by the blast should be loaded out before firing a subsequent shot.

DISCUSSION

W. J. GERMAN, Huntington, W. Va.—Although this paper is mostly about Illinois, in some ways it applies to other mining fields.

Regarding safety, much danger comes from one bad practice; namely, not seeing that all miners are equipped with suitable scrapers for getting the cuttings out of the borehole so that the explosive fast charges can be forced to the back of the hole. Often the explosive charge gets halfway back in the hole and when the shot fireman comes around, he tamps it where it is, which is the cause of much unsafe and unsatisfactory blasting.

In the last paragraph, the paper says that the coal should be loaded out from the breaker shot before the second shot is fired. That, of course, is not possible in Illinois because of the Illinois mining law, even though it is a great advantage and people in other states are enjoying that advantage.

Another bad practice that has been covered thoroughly in the paper is the drilling of holes before the places are undercut. This is going on generally in a great many states and something should be done to prevent it.

In regard to the question of cutting the ribs square, I find that many machine men prefer to sump in a little off to the side, because it eliminates about two settings up of the jack. Even though they continue to do this, the holes should be bored parallel with the center line of the place.

The practice of loading or charging boreholes before the places are cut is very bad, and is carried on extensively. It is dangerous because one shot may pull a great deal more coal than expected and the second shot is likely to be a blowout shot. I have seen many accidents caused by such practice.

The importance of the rate of detonation is a great argument among coal men and powder men. About three years ago there was quite a demand for permissible explosives that would eliminate the use of dynamite in the mines. A gelatin permissible was developed which proved satisfactory to take care of such work as ditching and rock tunneling and in blasting in coal seams where heavy rock partings are encountered. This explosive has been used in some coal blasting. Its speed is over 15,000 ft. per sec. and it has replaced explosives with a speed of only 7000 ft. per second.

Here is a letter from one of the companies handling the coal from these mines and an accompanying photograph (Fig. 7). The letter says:

"We enclose two kodak pictures showing lump coal from your operations consigned Tide 437 which was loaded into Barge Tennessee at Newport News on August 18.

"Coal of this size does considerable damage to the slide boxes and chutes on our coal piers and is also damaging vessels when loading, as well as delays loading on account of large lumps clogging up the chutes.

"Would it be practicable for you to reduce the size of such lumps when loading?"



FIG. 7.—COAL TOO LARGE FOR SAFE HANDLING.

DIRECTION OF HOLES

The section on use of flat holes is one of the high spots of Mr. Tiffany's paper. In my study of many blasting problems over the country, I find the workers will bore a hole pitching up just because it is easier to drill, where much better results would be obtained if they would bore the hole straight in on the level of the coal seam. This applies also in connection with mechanical loading where coal is cut on the top or in the center, leaving thick bottom benches to shoot. This is the reason: When you are boring on a pitch either up or down there are no two holes that go to the same depth; but if you bore from the bottom or even with the top, the auger can be put into the cut and marked with chalk to know the exact depth and all holes can be bored to the proper depth. It is a practice worth considering and I am glad it was brought out so forcibly in the paper.

Many misfires are caused by bad cables. The Paragon Electric Co. of Chicago has made an extensive study on what is required as a cable for blasting in coal mines and has developed an inexpensive cable in 100-ft. coils, rubber-covered, each wire insulated separately and high-grade rubber on the outside. About 60 per cent. of the complaints I have investigated on electric blasting caps and electric squibs are due to bad cables, and the information on these cables can be obtained from the Paragon Electric Co.

Just one more question: Is there any cause or reason why we have not changed over to the use of permissible explosives exclusively?

J. BARAB, Wilmington, Del.—The paper has brought out some important points, and most important, to my mind, are the recommendations for drilling holes properly and cutting and removing the cuttings. That may seem to be a matter of repetition, but it has not been brought home to a great many operators. I can not see the reason for cutting the coal if you are not going to remove the cuttings. The object is to give another free face, and if the cuttings are left in, there will not be another free face, so why spend that extra money?

As Mr. German says, the rate of detonation is something to which much thought can be devoted. The authors say: "Although these particular tests did not extend to other mines, the writers are of the opinion that there is one rate of detonation which will produce a maximum quantity of lump coal for any particular coal mine." It seems to me that that is a rather broad statement. It is pretty well known that the higher the rate, the greater the shattering effect, and while that may be open to some discussion on account of different conditions, as Mr. German pointed out, with an explosive of 15,000 ft. per sec. the operators had a complaint on account of too much lump coal. The important factor is to select the proper explosive, particularly as to strength and density, for the conditions under which you are working.

The authors say that in one case they had a saving of 10 per cent. from air-spacing. We have done some investigating on that line and have found that 10 per cent. is a rather conservative figure on the saving of explosives over the use of the same explosives without air-spacing. That seems reasonable, because the object of air-spacing is to increase the lump coal and reduce loss; it requires a certain amount of energy to shatter the coal and if the explosive is used in such a manner as to give only large lumps not so much energy will be needed.

E. H. SHRIVER, Nellis, W. Va.—The recommendation regarding permissible explosives seems to me entirely too broad. The use of permissible explosives is advocated to effect a greater safety of the men in the mines. The statistics of the U. S. Bureau of Mines show that far more men have been killed by falls of roof or coal than have been killed by catastrophes caused by explosives.

The mine with which I am connected carried out some tests last winter with the idea of using permissible explosives entirely as a safety proposition. We found that we would increase the hazards instead of decreasing them because our roof is so tender that the increased shattering effect of the permissible explosives over black powder brought about a greater hazard to the individual loader. As a result of these experiments, we determined on the use of compressed powder (or to use the trade name, Pellet Powder) fired electrically. We do not permit fuses. In mines where the content of gas is not so great as to cause the use of closed lights, or where the amount of coal dust is not so large as to render it dangerous from the flame standpoint, we believe that compressed powder is preferable to permissible.

The lump proposition did not affect our particular mines because our coal is crushed. The only reason we desire lump coal is for greater ease in cleaning. We did not adopt compressed powder for that reason. It was simply a safety measure.

J. KEELY, Kayford, W. Va.—Since the compressed or Pellet Powder has been mentioned, I want to tell some things that I have found out recently. We were using the permissible because in certain very wet places where we were center-cutting the coal and shooting the bottom, and had a great deal of moisture in the bottom, we found that we were shattering it clear out of reason with the permissible explosives. We adopted the Pellet Powder because it could be shot in damper places and take the place of Monibel, the trade name of the permissible we are using. We

found that we got a difference of about 9 per cent. in lump. Our miners shoot their own coal. They were offered a choice of powders and they very rapidly chose the compressed powder, which looked suspicious; recently I have found that they remove the paper covering of the Pellet Powder, put it back in the hole and pulverize it, tamping it very hard, and consequently "shoot the hell out of the coal," which is the miner's formula for a good shot.

E. McAULIFFE, Omaha, Nebr.—The reading of this paper impresses me again with the fact that it is not information we want so badly as it is the disposition to apply the information we have.

When I took charge of the Union Pacific Coal property in Wyoming five years ago, our non-gassy mines were not using permissible explosives. Going back over the annual safety reports made by outside mining engineers for a period of 15 years, I was impressed by the continuing recommendations made for the substitution of permissibles for black powder which were eventually carried out only to the extent of putting permissible explosives in the gaseous mines. I decided immediately that we would go over entire to permissible, and while in the beginning we met with a certain measure of opposition from the miners' union, we went ahead, and with the assistance of the powder men experimented with three or four brands, and thereafter we put permissible in 100 per cent. without serious friction or prejudice.

Ours are so-called captive mines, the output used exclusively by the railroad company, and much of the coal is reduced to 2½-in. size in order to meet the requirements of the locomotive stokers. Later we extended our mechanical loading program, and found in our subbituminous mines that the coal as shot was coming down in some instances in lumps too large for the Joy loading machine. Our men then attempted to reduce the size of the lumps without materially increasing the percentage of slack, and again with the help of the powder men we developed the theory of using a 1-in. permissible stick instead of the standard stick. That in turn meant a one-man electric drill; it meant specialists to drill the coal, place the shots, and subsequently to fire them. That helped our situation very materially.

The powder men were not eager to supply us with the smaller sized explosive but they did it and we are expanding that program. I would suggest, as a possible way out, more holes, smaller holes, holes that can be drilled perhaps more cheaply. The measure of fine material taken out of the 1-in. hole is much less, of course. Some of our men were actually making a hole 2.75 in. dia., creating 7.56 times the amount of drill cuttings produced by the 1-in. drill.

Research in Processes of Ore Deposition

BY WALDEMAR LINDGREN,* CAMBRIDGE, MASS.

(New York Meeting, February, 1928)

FIFTEEN years ago, in his presidential address before the Washington Academy of Sciences,¹ Alfred H. Brooks said: "Applied geology can only maintain its present high position by continuing the researches which advance the knowledge of basic principles. Future progress in applied geology depends on progress in pure geology."

Early in 1927, Dr. David White, then chairman of the Division of Geology and Geography of the National Research Council, appointed a committee "On the Processes of Ore Deposition," which was the first recognition by the Council of this branch of applied geology. This committee will not be able to solve the many complex problems involved; it can only clear the way, survey the field, state the problems and encourage systematic research. This short paper aims to outline and to analyze the present status of this kind of geological research, to ascertain the quality of the work and to suggest lines that might profitably be followed.

INCREASING DEMAND FOR GEOLOGISTS

Economic geology, or the art of geology, could develop only after the principles of geology and mineralogy had been established. Before that time it was merely a collection of trade rules accompanied by more or less wild speculations. Thus we observe the beginnings of economic geology in the period 1770-1820. Soon afterwards, the State Surveys were established in this country, with the realization that the natural resources should be investigated and described. Geology was hardly yet equal to the task of scientific research in relationships and genesis, however, and there followed a period of discouragement and stagnation. In the meanwhile, pure geology had again registered progress, and, based on its principles and also on the rapid mining development in the Cordilleran region, the present phase of the study of ore deposits began to take form. Raphael Pumpelly, S. F. Emmons and G. F. Becker were among the pioneers in this movement. The United States Geological Survey was organized. Ore deposits were scientifically described and studied along lines of structure and mineralogy; theories of genesis were advanced, not always permanent but based on sound induction.

* Chairman. Division of Geology and Geography, National Research Council.

¹ A. H. Brooks: Applied Geology. *Jnl. Wash. Acad. Sci.* (1912) 2, 19.

The geologist was coming into his own; the more progressive mining companies began to recognize his value, where formerly he had been at most a tolerated visitor. True, he made many egregious mistakes. Not so many, however, as the miner without any but local experience and no knowledge of science. He could not "see into the ground" but he could at least make an intelligent prognostication. In this stage, the mining engineer, in most cases, was the geologist. The science had not developed beyond the curriculum of the engineer.

Again, however, pure science forged ahead relentlessly. The origin of minerals was classified; new diagnostic methods were invented, all with a bearing on the miner's problems. The science began to grow beyond the ordinary education of the mining engineer; necessarily it became more specialized. It became obligatory for the geologist to familiarize himself with physical chemistry and metallography, with problems of sedimentation and colloidal solutions, with the phenomena of magmas and vulcanism. And lately the mining geologist is reaching out to the physical sciences for help that will enable him to predict the existence of orebodies or bodies of other useful minerals below the surface. These developments have still further emphasized the gulf between the mining engineer and the geologist. It is scarcely possible to dominate both fields. There are those who say that geological science has made no progress of late. Surrounded by trees, they fail to see how the forest has grown.

When the miner has an abundance of ore he gets along comfortably without the geologist, but when the ore resources grow slim, the geologist is called forthwith. Often he works at a disadvantage on account of a lack of knowledge of the details of the deposit as disclosed during the prosperous period. The value of the services of the geologist should not be exaggerated, but certainly he has a right to a place in the sun. Mining operations should be conducted by the mining engineer, but a trained geologist should superintend development operations. By this I mean not only trained in the various branches of geology and mineralogy, but also in the practical knowledge of mining. One of the urgent needs of many mines today is the recording of the geological data so that they may be available when needed—and they will be needed.

GEOLOGICAL DATA NEEDED

The data needed are of three kinds: (1) structural, involving a record of the attitude of the orebodies, thickness and grade of ores, attitude of faults and other structural features related to the orebody; (2) geological, involving the relations of the rocks about the ore deposit; (3) petrographical and mineralogical, including a study of the composition of the rocks, altered and unaltered, and of the ores as to ore minerals present and their relations.

At the present time a few large and progressive organizations employ a geological staff. Usually their observations regarding structure are complete and efficient; but regarding the study of the ores and rocks they are usually deficient. Quite naturally the structural elements have an urgent and immediate relation to mining operations; the usefulness of the mineralogical side of investigations is less striking. "Why investigate these matters?" we are asked. "The assays will tell us all we need to know." I shall not attempt to refute this argument in detail. Everyone should realize that if money is made from a natural product the operator must know all that can be ascertained about the product from a standpoint of modern science. If the ore contains palladium or vanadium, it is certainly money in the pocket of an operator not only to know the fact but to know in what minerals and in what combination these metals are present. It seems slightly ridiculous to advance such arguments in these days, but even yet the attitude of many mining companies is medieval, if not archaic in this respect. If people could be persuaded to be less strict utilitarians, other researches into the properties of their products might conceivably help them later on—or their children—to accumulate the desired wealth. The value of research for possible future benefits is recognized in the sciences of physics and chemistry today, and it may obtain in mining geology in the future.

Leaving this line of thought for the present, let us consider the activity in geology and in ore deposits in this continent of North America.

PRESENT STATUS OF RESEARCH IN PROCESSES OF ORE DEPOSITION

The activity of geological research may be rudely measured by the number of papers and books presented annually. In 1887, there were about 300 publications relating to the geology of North America; they rapidly grew to 1350 in 1903, and varied from 1200 to 1400 per annum between 1903 and 1917. Then followed a sudden drop to 1050, owing to war conditions. Since then, the number has risen again to about 1400 in 1926. The greater number of these publications have some bearing on economic geology; comparatively few deal with the pure research. Brooks estimated that 80 to 90 per cent. of the publications of the U. S. Geological Survey from 1903 to 1910 were related to applied geology, and the same condition holds good at the present time.

It is difficult to define what is meant by research. Evidently, the publication of statistical data of production is not research, unless generalizations are drawn from the figures presented. The routine description of a district containing mineral deposits is not necessarily research unless some attempt is made to deduce generalizations or to uncover new facts.

In order to obtain a rough estimate regarding the condition of research in ore deposition, I have attempted to analyze the bulletins issued by the

U. S. Geological Survey on the geological literature of North America for 1921-22, 1923-24, and 1925-26.²

PUBLICATIONS ON ORE DEPOSITS IN SIX-YEAR PERIOD 1921-26

Publications	1921-22	1923-24	1925-26
Total.....	2234	2577	2750*
Relating to ore deposits.....	257	222	358
Relating to ore deposits in the United States.....	175	145	213
Relating to ore deposits in Mexico, Central America and West Indies.....	10	9	34
Relating to ore deposits in Canada.....	72	68	111
On ore deposits, by the Canadian Geological Survey...	24	9	22
On ore deposits by Ontario Bureau of Mines.....	4	1	18
On ore deposits by the U. S. Geological Survey.....	36	10	17
On ore deposits by State Geological Surveys.....	6	22	15

* Approximate.

Strictly statistical papers have been excluded from this list. The publications not specifically accounted for include papers in part of research quality in *Economic Geology* and the journals of the mining and geological societies, also any short communications in the mining press that present anything of interest in processes of ore deposition.

Naturally, these papers differ greatly in quality; many are of little value, others embody much research; many of the latter are longer papers and memoirs of the Geological Surveys. Those of the U. S. Geological Survey stand first in research quality but are closely followed by the publications of the Geological Survey of Canada. Most of the actual research material finds publication in the scientific journals and there is, of course, considerable repetition, as authors often publish their most important results as separate articles in such journals.

While perhaps the greater number of important publications come from organized surveys, there is a considerable and growing contribution from the geological and mining departments of the universities. These papers embody scattered and not always well organized research. Among the institutions contributing are the Universities of Minnesota, Wisconsin, California, Yale, Harvard, Princeton, Columbia, Stanford and the Massachusetts Institute of Technology; the Universities of British Columbia, Toronto and McGill; and also the mining schools of Missouri and Colorado. From 3 to 10 papers are contributed per annum from

² J. M. [Nickles: Bibliography of North American Geology for 1921-22. *Bull.* 758, U. S. Geol. Survey (1924).

Same for 1923-24, *Bull.* 784 (1927).

Same for 1925-26, *Bull.* 802 (1928).

each of these institutions. Two stand out prominently as leaders: the University of Minnesota and the University of Toronto. In both, the research seems better organized than elsewhere. The University of Minnesota contributed 4 papers in 1921-22; 14 papers in 1923-24 and 11 in 1925-26, most of high quality. The University of Toronto contributed in the same periods, 16, 6 and 7 papers, most of them dealing with the mineralogy of ore deposits.

In looking over the publications for these six years one can not fail to be impressed with the number of able papers on the paleontology and the geology of coal and petroleum. Papers on processes of ore deposition make only one-tenth or one-eighth of the total number, and perhaps 20 per cent. of them contain little of importance in regard to this subject.

Another interesting feature is the great number of publications relating to Canadian ore deposits; they make up, roughly speaking, one-third of the total in each biennial period. Undoubtedly this indicates a much more lively interest in the subject than is found in the United States. When we consider that the population of Canada is less than one-tenth that of the United States and that the value of the metallic production in the States is more than ten times as large as that of Canada, the relation becomes still more striking. Either Canada spends too much effort and money in the investigation of its metalliferous deposits or the people of the United States spend too little. Probably the truth lies in the latter statement.

WHAT CONSTITUTES "RESEARCH?"

G. K. Gilbert recognized two fields of geological research, the one embracing the study of local problems of stratigraphy, structure, ore deposits, etc., the other the general problems of geologic philosophy. Both, he held, may yield results of high economic value. On the other hand, it is generally agreed that compilation of data, including statistics, administration and the routine determination of minerals and rocks, should not be considered as research. It is thus evident that the activities of geological surveys to a large degree fall within the definition of research.

On the other hand, research may be considered as the search for something new, and if the structure, ore deposits, etc., described present nothing new, nothing that will aid in the development of the science, such work can not be classed as a high order of research. Even if new data are described—like new minerals in an ore deposit—the next stage of correlating these data, of reaching new viewpoints, is often neglected. Ransome has truly said that in any early stage of scientific inquiry copiousness of description and futility in hypotheses are dominant, and that such descriptive and unverified hypotheses even now preponderate in the literature of ore deposits.

ACTIVITY OF THE SURVEYS

The U. S. Geological Survey

The Federal Geological Survey stands first in regard to literature and research of ore deposits. It is not necessary to detail its achievements along these lines. In the last 40 years it has accomplished the investigation of a majority of the ore deposits of the country and summaries of the occurrences in many states and regions. Doubtless it has done all that was possible under the circumstances, for even the Federal Survey feels that it is hardly in its province to take up problems of ore deposition which naturally must involve a great deal of laboratory work.³

The Geological Survey has been still further constricted by decreasing appropriations, by failure of the appropriating and departmental officers to realize that a geologist is a scientific man who needs something more in the way of equipment than his notebook, a desk and a pencil. It is not generally realized that the Geological Survey serves not only the mining industry but also, through its topographic mapping, its hydrographic work and its land classification, wide industrial and engineering circles. It would seem as if the country hardly appreciated its services. The control and regulation of rivers, the investigation of earthquakes, the soil surveys, the geophysical methods of discovering bodies of ore or other useful substances—all of these subjects would seem to belong, in part or wholly, to the province of the Survey. It may be said that most other countries treat their geological surveys better than we do and have a better appreciation of their value.

These views recently were sharply expressed in a resolution adopted at a meeting of the Columbia Section of the American Institute of Mining and Metallurgical Engineers, in which it was pointed out that the mining and allied industries suffer severely through lack of sufficient appropriations to the Bureau of Mines and the U. S. Geological Survey. It was shown that with a value of production of about \$13,000,000,000 for agriculture and \$6,000,000,000 for the mineral industry, the appropriation for the former (to the Department of Agriculture) is about \$128,000,000 per annum and that for the Bureau of Mines and the Geological Survey together about \$4,700,000. It is estimated that the Department of Agriculture spends about \$10,000,000 annually on investigations which may be classified as "research," while the U. S. Geological Survey probably devotes much less than \$100,000 to similar work in all branches of geology. It may be imagination, but one has a feeling that the Geo-

³ Much research of high quality has been carried on in this direction by individual geologists and chemists. As a rule, however, when the particular report on a given district is finished, the author ceases his work. The greater generalizations, which can only be gained by comparative study of many similar districts, are usually left in abeyance and the investigation of the next area or district begins.

logical Survey presents plans for scientific research with an apology for their less evident economic utility. There is, of course, a well-defined reason for this: agriculture is carried on by millions of small, financially weak operators while in the metal industry there are a great many corporations of great financial strength. However, even in the latter industry there are also many hundred of thousands of small operators who are exactly in the position of the farmers. Elaborate comments on this anomalous condition are unnecessary. It certainly appears that the agricultural industry is able to obtain from Congress almost all that is asked for while the mineral industries do not receive a corresponding consideration.

For a number of years the Geological Survey has been allotted from \$1,600,000 to \$1,900,000 per annum. Of this from \$300,000 to \$500,000 is allotted for topographic mapping. Other items are for printing, water resources, classification of public lands, etc. To geology is allotted about \$325,000; this amount has remained about constant for many years, but since salaries and expenses have greatly increased in the last ten years, it is apparent that less work is obtainable for this amount now than before. The sum of \$325,000 covers all kinds of geologic work, much of it not directly related to the metal industry. It may be estimated that geological investigation of the metal industry amounts now to about \$50,000 to \$70,000 per annum. This is the amount devoted to the investigation of the ore deposits of an industry which produces \$1,400,000,000 worth of refined metals per year. The amount actually expended to investigate the processes of ore deposition is naturally only a fraction of \$50,000 or \$70,000.

Undoubtedly, the Bureau of Mines and the Geological Survey have accomplished an enormous amount of work with these limited means, but this is scarcely an argument for continued low or reduced appropriations.

The State Surveys

Much work is also done in the investigation of metalliferous deposits by about 30 State Surveys. It is difficult to state the total amount expended by these; probably it is not much more than \$350,000 per annum, and of this naturally only a small fraction is devoted to processes of ore deposition. The State Surveys even more than the federal Survey are subject to limitations regarding research work of the higher order. Quite naturally, perhaps, the state authorities balk at expending money on anything but classification and description of ore deposits. Nevertheless, some of them—for instance, the Surveys of New York, Maryland and Missouri—have shown an appreciation of the broader aspects of economic geology. Tedious detailed descriptions are often an unavoidable fault of the State Survey reports.

The Canadian Surveys

For the current year the Canadian Government has allotted \$233,400 for the Mines Branch and \$335,000 for the Geological Survey. The former sum includes work for the publication of memoirs of special substances such as molybdenum, graphite, asbestos, etc., which contain a certain amount of research work. Of the whole, however, but a small proportion of the moneys for the Mines Branch goes into research of processes of ore deposition. The allotment for the Geological Survey provides for some topographic work, publications, museum expenses, etc., but from the fact that about 23 field parties were working last year in metalliferous area, and from the character of the reports, it seems clear that much attention was given to ore deposits. Exact figures are not obtainable but the guess may be hazarded that the amount spent is not less than \$80,000, a sum materially larger than that devoted to the same object by the U. S. Geological Survey.

The next largest organization is the Department of Mines of Ontario. Many interesting and valuable reports are received annually from this source and much attention is given to ore deposits. For the year ending Oct. 31, 1926, the appropriation for geology, mining and topography amounted to about \$90,000. Most of the work relates to metalliferous deposits. While it is not practicable to segregate the amounts used for this purpose, they may be estimated at not less than \$50,000, or about the same amount that is expended for similar purposes by the U. S. Geological Survey.

CARNEGIE INSTITUTION GEOPHYSICAL LABORATORY

There remains to mention one institution whose work has always been of the very highest quality and from which much may be expected in the future—the Geophysical Laboratory of the Carnegie Institution at Washington. The geochemical data that we owe to it are indirectly of the greatest importance for ore research, and the investigations on hot springs and spring deposits as well as the reports on the supergene sulfides and the sulfides of copper, iron and zinc have been of such preeminent research value that we may hope for extremely valuable results if the Laboratory should see its way to undertake more extensive work on ore deposition.

INTEREST OF THE MINING INDUSTRY IN GEOLOGY

An industry which in this country produces metals to the annual value of \$1,400,000,000 must be presumed to be deeply interested in the occurrence, origin and discovery of the deposits from which it produces this enormous wealth. No doubt the metallurgical industry has

expended large amounts on research in the reduction of ores and in the composition and behavior of metals and alloys. This branch of science is certainly progressive and eager to grasp at the benefits of research work, but what about the mining industry? The answer must be that the mining industry has taken little or no interest in the scientific study of deposits, beyond the utilization of such data as may be supplied by the official Surveys. It has put little or no money in the investigation of processes of ore deposition or in creating scholarships for such research.

True, some of the mining companies maintain geological staffs but only to investigate their own problems, and many employ geologists when the ore resources begin to dwindle, but in processes of ore deposition they evince practically no interest. A few mining schools have been endowed by individuals, who obtained their wealth from the mines, but the activities of these schools are almost entirely in the direction of mining methods and metallurgy.

There are some exceptions, of course. The Copper Producers Association, for instance, some years ago authorized the Secondary Sulfide Investigation conducted by Professor Graton. Among the results of this study are the invaluable data on copper sulfides by the Geophysical Laboratory referred to above. The Calumet & Hecla Co. conducted a thorough investigation of its copper deposits and generously allowed this investigation to be published. The Phelps-Dodge, the Anaconda interests and other companies have also permitted the publication of certain researches conducted by them. But on the whole the indictment probably will stand.

This condition is by no means confined to the United States. I understand that the gold-producing companies of the Witwatersrand district employed no geologist continuously till 1915 and that since then the only one so employed has been Dr. E. T. Mellor. This whole subject has been interestingly treated by Dr. H. C. Boydell,⁴ of the Institute of Technology, in a paper read before the Institution of Mining and Metallurgy (London).

The failure to make general use of geological control and to devote money to the investigation of processes of ore deposition may be caused by various conditions, such as an abundance of rich ore, or the simplicity of geological structure or unsatisfactory experience with poorly trained geologists. But fundamentally, I believe, it is caused by a deep-seated mistrust of the efficiency of geological methods. The services of geology may be admitted for structural problems and, in fact, most mining geologists working for the industry are engaged in such investigations. The inquiry into the composition of the ores is usually neglected; few mining

⁴ H. C. Boydell: *Economic Geology and the Mining Industry*. *Trans. Inst. Min. and Met.* (London) 35th Session (1925-26) 76-101.

concerns know all they should know about the chemistry and mineralogy of their ores. The question as to the origin of the ores is held to be purely academical, and of no practical importance; there is little discrimination between wild hypotheses and sound theories; it is held that "the geologist cannot see any further in the ground than the miner" in oblivion of the fact that the mind trained in mineralogy and structure is in far better position to diagnose the situation and discern the probabilities than the untrained person.

The situation is better than in the past but it is bad enough and all efforts should be made to correct it and to make the industry realize that research is necessary. The era of rich surface ores is passing; ores of lower grade are being utilized. No longer are there many boldly outcropping ore deposits which may be attacked with little thought as to composition or origin. As Boydell has said, "Individual deposits may be exhausted but mining must go on."

WHO SHOULD DO THE WORK?

Granting that physical and chemical research of the highest order is necessary and that smaller mining companies are rarely in position to maintain a research laboratory, who should conduct these investigations into the nature and origin of ore deposits? Many will unhesitatingly reply, "The Government," meaning the federal surveys; others may say "The Universities;" still others may say "Central laboratories like the Geophysical Laboratory of Washington." One may point to the brilliant results obtained by the research laboratories of the General Electric Co. and the American Telephone & Telegraph Co. and argue that large corporations should undertake the work.

There are certain objections to having the federal government undertake the work. It would be hampered to some extent by regulations, appropriations and red tape. Still, I am sure the Geological Survey could be depended upon for eminently successful work. Some kind of financial cooperation might be established similar to that existing between the Bureau of Mines and the steel industry at the Pittsburgh experiment station. The research laboratory of a large company is organized mainly for the benefit of that particular company. Certainly, the universities will continue to lead the research as they have done in the past. But it seems to be a better plan to have this work done in a central laboratory, independent or connected with some nation-wide organization like the Research Council of the Academy of Sciences, which would disburse the funds. It might be called the "National Laboratory for Ore Research." The choice would lie between the U. S. Geological Survey, the Carnegie Geophysical Laboratory and a "National Laboratory for Ore Research." In any case good results could be depended upon.

WHO WOULD PAY FOR THIS WORK?

Should the expense be paid by the "Government," that is by the people of the United States, or should the money be provided by the mining industry? Dr. Boydell unhesitatingly answers, "By the industry," and I think he is right. The same causes which have prevented generous appropriations from Congress in the past will be operating in the future. If the mining industry desires this work to be done it had better provide for it, instead of waiting for uncertain action by Congress. Annually it extracts from the ground metals to the value of nearly one and one-half billion dollars, and the expenditure of one ten-thousandth part of this sum for the purpose of obtaining more knowledge of the materials which furnish this great wealth would not seem to be exorbitant. Probably some such plan would have been accepted years ago if the industry had been adequately organized and if it had realized the value of modern research.

The metallurgical industry, quite as much as the mining industry, needs adequate knowledge about ores. The time has passed, though many may not realize it, when the assayer was the only arbiter and investigator of metallic raw materials. The metallurgical concerns have given much attention to their products, but have done little research into the nature of the ores.

For action in the matter: We have three eminent organizations whose duty it might well be to start the good work: they are the American Institute of Mining and Metallurgical Engineers, the Mining and Metallurgical Society of America and the Society of Economic Geologists. Certainly the National Research Council is at all times willing and anxious to lend its help to meritorious plans of investigation.

THE WORK TO BE DONE

As Ransome has said, there has been an abundance of general description of ore deposits, coupled with a tendency to indiscriminate theorizing. We need, as the same author has said, a greater accuracy in description and the establishment of fundamental physical and chemical data by experimental work.

In the study of the structure of ore deposits there is much to be done; there are plenty of individual descriptions but little comparative analysis of the features common to many deposits. No satisfactory structural study of the "lenticular veins" is available, for instance. There is still much uncertainty as to the manner of filling and replacement in fissure veins. The questions relating to fibrous veins and to inclusions of rocks in vein material need more attention.

Comparative studies are needed of certain classes of veins—for instance, gold quartz veins—the common features, the relation of depth

to structure, the relation of composition of the ores to structure and depth. We have much material but no master mind capable of analysis and correlation. To what depth may we expect these veins to continue? What is the influence of pressure and temperature on the precipitation of ores? Surely these questions are of great importance.

What are the common features of epithermal veins? Is there justification for considering them as the upper continuation of batholithic veins, or are they derived from different sources? This also is a question of great moment to those who would risk money in following them in depth.

A solution of many of these problems might be obtained from an analysis of the literature coupled with field studies; many are easily accessible to investigators in Surveys and Universities, but few men have taken the pains to follow them out.

Other lines of research can be undertaken only with chemical and physical aid, and these properly belong to the work of research laboratories. A few of the more important are enumerated here.

In connection with placer deposits, there is wanted a metallographic investigation of native gold and better knowledge of the relative rate of solution of silver and gold by natural waters, with consideration of the possibility of colloidal transportation.

In deposits of sedimentary origin, the conditions under which iron, manganese, copper and zinc are precipitated deserve more attention. It is known that zinc is contained in many sedimentary beds; for instance, in the Jurassic iron ores of England and in limestones of the Appalachian belt. Under what conditions would copper be precipitated in sedimentary beds from dilute solutions? What are the minerals formed and by what means—adsorption or chemical action? Why are hematite-silicate iron ores not formed in the seas of the present period? What is the origin of the sedimentary bauxite deposits?

In the deposits formed by processes of weathering, there are many aspects that will bear much more research, such as the decomposition of minerals, the development of the clay minerals, and of bauxite from the latter. Recent investigations suggest that the hydrogen-ion concentration is of great importance in these processes. In the future much of our iron ore will be taken from such deposits. Just what are the conditions for the development of limonite or of hematite? Many of these deposits form a potential source of nickel, not much utilized as yet. What conditions favor the concentration of nickel?

The processes which have formed our enormous iron deposits of the Lake Superior region will need additional examination to make possible the utilization of the low-grade ores.

In the future, when the richer copper ores are exhausted, the low-grade ores so widely disseminated in the sandstones of the Southwest

will no doubt receive attention. At the present time the conditions governing these deposits are little known; opinions as to origin are far apart. What minerals do they contain, and may we expect their continuation in depth?

Closely allied to these are the vanadium and uranium-bearing beds of the same sandstones. Just now they are eclipsed by richer ores from the Congo, but undoubtedly they will again assume importance. Whence came the vanadium and the uranium-radium? The present minerals are in large part secondary. How were they concentrated? What agencies caused precipitation of these metals? What part does organic matter play? Do the ores continue in depth?

An enormous amount of discussion and theorizing has been carried on about the lead and zinc deposits which at shallow depth have yielded such astonishing quantities of metals in the Mississippi valley. Are they of shallow, meteoric origin or may these ores be sought for in depth by drilling? If they really are of magmatic origin it would seem that there might be such a chance. Many excellent researches have been started to elucidate their origin but none have gone far enough. The whole question of the dissolving power of natural waters and the reactions of precipitation needs more work.

When we come to the ore deposits of admitted magmatic affiliations, the problems multiply and become more complex. Among the fundamental data required may be listed the behavior of silica, water and metallic sulfides under various conditions of temperature and pressure. Is silica always precipitated from colloidal solutions? Did the large crystals often found develop from such colloidal solutions? Were these colloidal solutions always a differentiation product of the magma?

The next question is in what form the sulfides existed in the water—for instance, the enormously predominant pyrite. And, further, by what means were they precipitated from their solution?

And after that comes the momentous problem as to why the sulfides are precipitated in a certain, more or less constant order. Is the zonal theory really based on physicochemical laws—if so, what laws? The field evidence is contradictory in part. Why are the arsenides and pyrite always the earliest, why the silver and silver-lead sulfantimonides always the latest minerals? What are the relations of tellurides to gold?

In the pyrometasomatic (contact metamorphic) deposits the calcium-magnesium-iron silicates develop and are rightly assumed to indicate a higher temperature. In certain lode deposits they do not form in the limestone, but cryptocrystalline silica develops instead. At what temperature do these silicates form? The system silica-calcium carbonate-water, for one, has never been investigated. It would seem that experimental work might well throw light on the origin of these contact metamorphic deposits.

The conditions under which the pegmatite dikes were formed have been investigated in part by the Geophysical Laboratory and we may expect further results from this authoritative source. On the other hand, the products of differentiation which we designate as sulfide melts have received scant experimental investigations. The Sudbury nickel deposits are supposed to be formed by such differentiation and it would be of the highest importance to discover similar ores in this country. The characters of sulfide melts, their temperature, their relation to mineralizers such as water or phosphorus, their action upon rocks—all these problems await competent study. The points of similarity of these melts with metals, mattes and alloys, the peculiar processes of “unmixing” or separating out of new compounds from a still hot originally formed product—these also deserve investigation.

It would be wearisome to expand these suggestions for research on ore deposition, but they show plainly the great need of investigations in all parts of the field. Some of them will be of immediate value; others may lead to valuable results which we can not now accurately forecast. Is it not well to abandon medieval views and realize that research means knowledge, and knowledge means power?

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DISCUSSION

E. L. BRUCE, Kingston, Ont.—Perhaps Dr. Lindgren has given you the impression that Canada is rather better off in the matter of research in economic geology than is the United States, but I am afraid the figures are not quite so favorable as they might appear. A great deal of the money spent in so-called economic geology is spent chiefly in description and not in productive work, as it should be. In the university with which I am connected we have been trying during the last two years to establish a research chair in economic geology. It looked very easy to raise \$100,000 endow-

ment for that purpose. We went to one of the mining companies which has produced millions in profits and they offered us \$1000. That is the state of things in Canada, and if we are any more favorably located than you are, you are certainly very badly off.

G. F. LOUGHLIN, Washington, D. C.—I think the mining interests of Colorado can set a fair example, not a complete one but a good beginning. They have been assessed according to the production of metal and have devoted a portion of that assessment to the geological investigations in their state, particularly to ore hunting. The U. S. Geological Survey has given them such results as it could give quickly during the last two years, and they realize that it takes time to prepare a complete report; also, that certain conclusions of ultimate practical importance can not be reached without a certain amount of research.

They have given us a free hand to go ahead. They will put up the money or pay the bills but they will let us plan the work. As research matters come up we devote what we think is a fair proportion of the time to those things.

If the mining interests of the whole country that are assessed according to production could arrange it so that a minor fraction of that assessment could be devoted to research in problems of ore deposition or problems of mineral deposition as a whole, there would be a fund adequate for the development of a research laboratory; independent, or if preferable, associated with the Carnegie Geophysical Laboratory at Washington or the U. S. Geological Survey, which already have a great many facilities for further research. An adequate fund for supporting such a laboratory could thus be raised and it could go to the research as a main issue and not as a minor or side issue.

C. P. BERKEY, New York, N. Y.—Conditions are becoming a little more promising. Perhaps a growing appreciation of our own shortcomings is a step in the right direction. It ought to be possible, under Dr. Lindgren's leadership, to get recognition of the importance of some of the well-known ore deposits with support for better studies of them. Usually if one asks the owners for opportunity to study their deposit it is looked upon as a bid for a job, and generally one does not even get a chance to begin.

Perhaps Mr. Lindgren, backed by the economic geologists, could present such a request with better effect. There are deposits of which the study promises more to pure science theory than to the solution of any immediate problem in handling the mine. A study may not be warranted for its practical bearing alone. We ought to be able to arouse interest in the case as a contribution to the science. But clearly we have not yet reached a satisfactory common understanding with our natural supporters and an occasional priceless opportunity passes.

R. H. SALES, Butte, Mont.—Dr. Lindgren's paper is timely and valuable. In our consideration of the attitude of the mining industry toward the economic geologist we must not overlook the fact that in applying geology to mining we do not often get results that are widely applicable. In this respect we differ greatly from those engaged in research work in certain other phases of the mining business. For example, if a metallurgist working on flotation makes a discovery, it immediately becomes helpful and valuable to every reduction plant in the world working on similar ores. Similarly, an improvement in a rock drill or blasting powder means a saving to every mine operator.

The economic geologist, on the other hand, rarely finds two mining districts with identical geological conditions. Each district has its own problem. Successful research in one district may or may not result in discoveries applicable to others. It is, therefore, rather difficult to plan research work in geology that will be beneficial

to the mining industry as a whole. Mining companies, realizing this situation, are perhaps skeptical about the claims and promises of the geologist.

Then, too, the economic geologist works constantly on theory. Orebodies are never fully outlined by mine openings or drill holes. His theory cannot be demonstrated by anything less than expensive mine exploration, and too frequently it proves correct in principle but fails as to quantity of ore. The mine operator can sample and measure ore already developed but the geologist is asked for opinions concerning that which he cannot sample or measure. There are, therefore, good reasons why his reports and opinions are so commonly written in conservative fashion, and consequently not to the liking of the promoter.

Here is where the geophysicist comes into the picture. Recognizing the geologists' difficulty in seeing into the ground or making quantitative determinations, the geophysicist is endeavoring to take a short cut to the orebody regardless of the geological condition surrounding it. I do not mean by this, however, that the geophysicist does not make careful study of geological conditions in connection with his work, because he is, in fact, an economic geologist who realizes the limitations imposed by present-day knowledge of ore deposits.

I think I can state with confidence that the mining industry is ready and anxious to give financial aid to research studies covering ore deposits. This being true, it is squarely up to the economic geologist to lay out the work and sell the idea to the operators, and this can be done if it offers a reasonable chance of success.

I know of nothing that more clearly demonstrates the mine operators' attitude toward economic geology than Dr. Lindgren's observation that the morale in the departments of geology in our colleges and universities is low because of the fact that the best of the younger instructors in economic geology are being snatched away by big mining companies. If Dr. Lindgren is correct, it is plain that the universities are furnishing not enough qualified geologists rather than too many.

Without question there is an urgent need of research work on ore deposits. One of the first requirements, before financial aid is sought from mine operators, is a well thought out plan of research work which will best meet the needs of the mining industry.

W. LINDGREN.—There are two things involved, it seems to me. One is the idea of a geologist making a report for a mining company for specific purposes. Naturally he cannot go into unlimited scientific detail. He will try to supply the information needed as best he can, and it has been said several times that he cannot see into the ground, but he can draw analogies from similar deposits. There are a great many deposits which are very similar; he can point out the probabilities at least, and he should be able to do that better than any other man.

Contrasted with that is an entirely different question; that is, to get to the fundamental problems of ore deposition. There is nothing in view there that promises returns in money right away. You can say that generally speaking the ores came from below. But I ask you, where would we be if all other scientists had stopped at generalities or all the other industries, we will say? What have they done? They have gone ahead and engaged chemists and physicists to start on lines of investigation that perhaps at first did not seem to be promising. Nobody knew exactly where they were going to lead. When they got through they knew something and they found it did lead somewhere.

I do not expect individual mining geologists to undertake such things. That must be done by central laboratories, and must be undertaken with the clear recognition that immediate returns in dollars and cents are not in view but are probable in the future.

So I think we have to cut away from the idea of immediate profit in dollars and cents. I ask you to look around at the other industries and see what they are doing and if they are not proceeding on that basis.

MEMBER.—There is the General Electric Co., for instance.

W. LINDGREN.—Yes, that is a good example. The geophysical prospecting is a splendid thing and is a wonderful line of new promising work. I want to point out to you one thing about the geophysical prospecting; it cannot get anywhere without geologists. I say that without qualifications. To interpret the result will need a geologist.

G. O. SMITH, Washington, D. C.—Of course, I think the majority of us, without partaking in this discussion, endorse all that Mr. Lindgren has said. First of all, we must recognize the need for this fundamental research.

I like to think of the work of the geologist in connection with our great metal industry as in the nature of taking out insurance. The fundamental research program, such as Mr. Lindgren has outlined, is of the nature of a long-term endowment policy. We will get the return, even though it may be at some distant time in the future. We have confidence that the return will come. There is also this present need of something immediate and tangible in the way of results, which is quite distinct from the fundamental work. The two, however, may and should go along together.

The latter kind of short-term insurance is the type that I think the State of Colorado, is taking on. And we can hardly think of it as a tax. What we pay for this insurance is a very small item indeed if we put it in terms of value of the output of the industry.

In discussing this subject in our meetings of the section delegates during the last two days, we noted the unfortunate fact that the amount of funds available for federal geological work has been decreasing rather than increasing in the postwar period, whereas the volume of production of our mines has increased rapidly, something like 40 per cent. increase since 1919, this being the period in which we are living and working, as I expressed it in speaking to the delegates. That 40 per cent. increase should have something similar by way of increase in the money available for searching out the conditions favorable for the discovery of new ore deposits, to replace those we are depleting at this accelerated rate.

I think in 1919, where we started the curves used in this discussion, the amount of money that was being spent in Federal geological work represented a tax, if you wish to think of it in that way, of less than one cent to every \$100 of mineral product. That is hardly noticeable and we know that it is utterly inadequate if it is to provide insurance for the future prosperity of the mineral industry.

There is so much to be done and so little to do with. Naturally, those of us who have this matter at heart welcome all of this discussion, because we feel that it will bring home to some mining companies, at least, the practical need and value of this type of work.

I think Mr. Locke was one of the first to use the term "ore finding," which is a significant word and rather inspiring and helpful as a word, and I am glad to say that our Federal geologists are using it in connection with the work in Colorado. They have the definite purpose of helping to insure the future metal industry of that state by going directly and with a fixed purpose after new orebodies.

I have not been at liberty to attend these recent sessions on the subject of geophysical methods, although I have attended some of the preliminary discussions, and when I listen to the reports of progress or read those reports, I am both encouraged and discouraged. It is a new method and one of promise, but I quite agree with Dr. Lindgren that it does not do away with geology. It is strong medicine, but I

think you will find you need a preliminary dose of geology, and then you need a "chaser" of geology.

We have had considerable experience in using airplane photography in our Federal topographic mapping. At first we thought there was a possibility of its cheapening and expediting the making of topographic maps, even if it might not do away with all the work on the ground. We find, of course, it is not doing away with the work on the ground. It is not expediting that work materially, and it is not cheapening it, but it is benefiting the work in that we are making a much better map, and I think that is what geophysics will do. We will get more results but we will have to use the old methods as well and the better results will come by way of supplementing the work on the ground and under the ground in this case.

We must agree with the premises as stated in Dr. Lindgren's paper, and we must follow him in his conclusions. There is no other way out. We need more fundamental research and we need more of the application of that work in the field by the men working in individual districts or in the employ of the large companies on the individual properties.

Discussion and publicity will help us. Geology has sold itself to the oil industry to such an extent that I doubt if there is at present any oil company without its geologist or its corps of geologists.

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The State and Density of Solutions Depositing Metalliferous Veins

BY WILLIAM H. EMMONS,* MINNEAPOLIS, MINN.

(New York Meeting, February, 1928)

THE problem of the metalliferous veins has always been an outstanding one in the science of ore deposits. In recent years interest has been stimulated by Spurr, who maintains that veins have consolidated from magmas that have densities as great as the densities of ordinary rocks.

Lindgren published a noteworthy paper¹ in 1926, in which he supports the theory that veins are deposited chiefly by hot waters. Lindgren's paper was discussed by Singewald, Spurr, Kemp, Anderson, Colony, Bateman, Porter and others. The discussion, in the main, was centered on the interpretation of fragments that are included in veins. This paper also treats the problems of fragments in veins and includes a reply to a review by Spurr² of a bulletin by Larsen and myself³ published by the United States Geological Survey on the geology and ore deposits of Creede, Colorado.

Mineral veins containing fragments of the wall rock are common. At many places the fragments are completely isolated, one from the other, and are entirely surrounded by vein matter. It has been stated that the vein matter around the fragments was heavier than the fragments and that the latter were floated in a magma. J. E. Spurr, who is the leading advocate of this hypothesis,⁴ urges that normal mineral veins were deposited by solutions heavy enough to float fragments of the country rock, and it is mainly on this relation that he bases his theory that veins are deposited by solidification of magmas.

If it can be shown that vein-forming solutions have a specific gravity greater than rock (2.77), one is justified in regarding them as true magmatic solutions. It is shown later in this paper, however, that at Lead, S. D., water not more highly concentrated than ordinary drinking water is depositing vein matter around fragments of schist and of steel in such a manner that the fragments are completely isolated and completely surrounded by the mineral cement.

* Director, Minnesota Geological Survey.

¹ W. Lindgren: *Magmas, Dikes and Veins. Trans.* (1926) 74, 71.

² J. E. Spurr: *The Filling of Fissure Veins. Review of a paper by W. H. Emmons and E. S. Larsen on Creede, Colo. Engng. & Min. Jnl.* (1923) 116, 329.

³ W. H. Emmons and E. S. Larsen: *Geology and Ore Deposits of Creede District, Colorado. U. S. Geol. Survey Bull.* 718 (1923).

⁴ J. E. Spurr: *The Ore Magmas. New York, 1923. McGraw-Hill Book Co., Inc.*

MAGMAS

We may consider two kinds of magmas; one a very thin, heavy liquid mass; the other, a heavy viscous mass. In the thin, heavy, liquid solu-

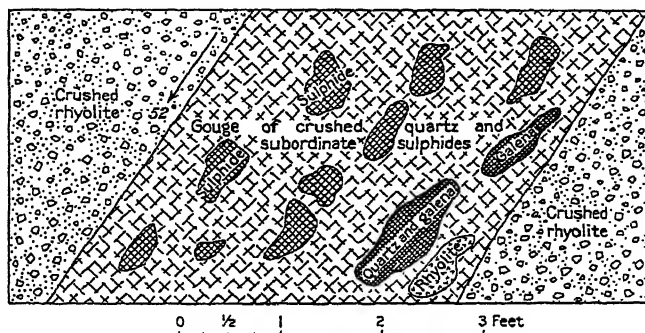


FIG. 1.—SECTION OF AMETHYST VEIN, AMETHYST MINE, LEVEL 6300 FT. SOUTH OF SHAFT, LOOKING NORTH.

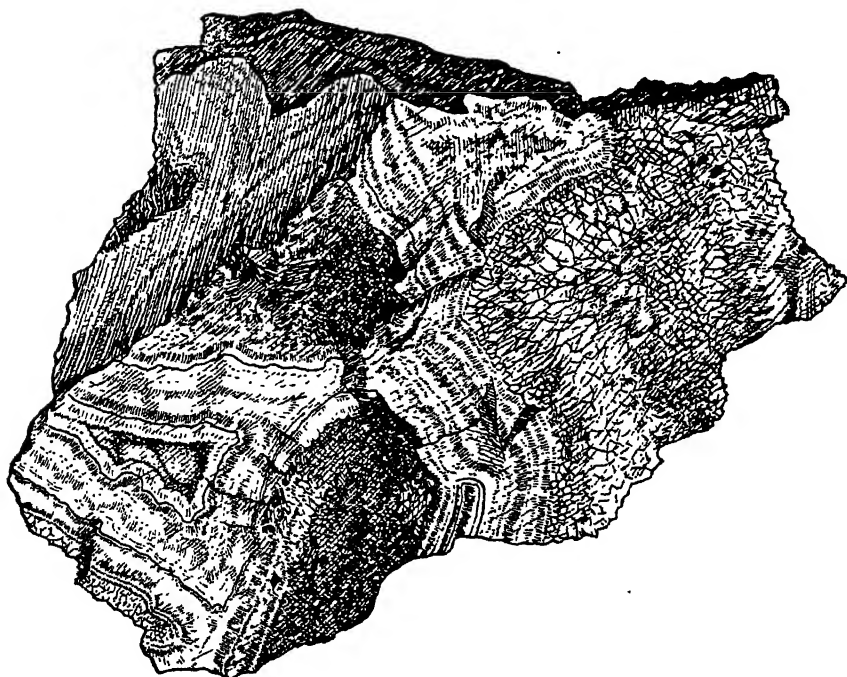


FIG. 2.—BRECCIATED ORE CEMENTED BY QUARTZ, AMETHYST MINE. THE DARK AREAS ARE FRAGMENTS OF SPHALERITE, GALENA AND QUARTZ. THESE ARE SURROUNDED BY BANDED MILKY QUARTZ ON WHICH AMETHYSTINE QUARTZ IS DEPOSITED. A VUG IS SHOWN IN LOWER LEFT CORNER.

tion few fragments would be found in the solution after it had solidified. Those heavier than the magma would sink and those lighter than the

magma would rise. Only the fragments with the same density as the magma would remain unsupported within the liquid mass. In the Amethyst vein at Creede, Colo., which, according to Spurr, was formed by cooling of an ore magma,⁵ there are found near together fragments of galena ore and of much lighter rhyolite in the same exposure of the Amethyst vein (Fig. 1). The vein dips 52° and a fragment of rhyolite is found below a fragment of galena ore. In a liquid solution heavy enough to float the galena ore, the rhyolite fragment would rise to the hanging wall. There is no evidence whatever of a separation due to different densities of the incorporated fragments.⁶

In a viscous solution which is moving, fragments of various densities may be incorporated in the mass. A rhyolite flow breccia will include

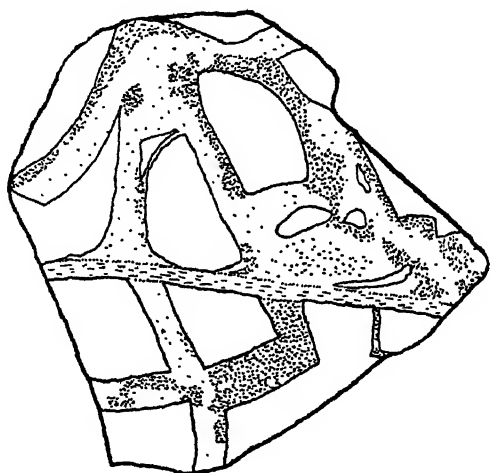


FIG. 3.—SPECIMEN OF ORE FROM PHILIPSBURG, MONT. FRAGMENTS OF RHODOCHROSITE (WHITE) ARE CEMENTED BY SPHALERITE AND QUARTZ (STIPPLED). THE LATTER IS CUT BY A VEIN OF QUARTZ WITH A LITTLE SPHALERITE. SOME OF THE RHODOCHROSITE FRAGMENTS WOULD FIT IF BROUGHT INTO JUXTAPOSITION.

fragments of rhyolite and of basalt near together. Such a rock after solidification will show flow structures or other evidences of viscosity.

Cooling of rock generally results in a decrease of volume, as is shown by the formation of cooling cracks after solidification has begun. If a magma cools to form an igneous rock, the rock should be heavier than the magma. In normal veins a gangue of pure quartz (sp. gr. 2.65) often surrounds fragments of heavy ore. This is illustrated by Fig. 2, a drawing of ore from the Amethyst vein at Creede. In other deposits, an ore of sphalerite (sp. gr. 4.1) and quartz surrounds fragments of rhodochrosite (sp. gr. 3.5). This is illustrated by Fig. 3.

⁵ J. E. Spurr: The Filling of Fissure Veins. Review of a paper by W. H. Emmons and E. S. Larsen on Creede, Colo. *Engng. & Min. Jnl.* (1923) 116, 329.

⁶ *Op. cit.*, 115.

In the Centennial range of northeastern Nevada, about a mile southeast of Edgemont, there is a great vein of barren quartz several feet wide which contains a few fragments of limestone completely isolated. The specific gravity of calcite is 2.72 and that of quartz is 2.65. No sulfides or other minerals are included in the quartz. There does not seem to be any constant relation between the specific gravity of vein stuff and that of the included fragments. Some of the latter are heavier and others are lighter than the cementing vein stuff.

COLLOIDS

In certain vein-forming solutions, the minerals are probably carried in the colloidal state. A solution of this character may or may not set as a jelly. Before such a solution sets, fragments that are heavier than the solution will sink and those that are lighter will rise. The gel would probably be as dense or denser than the solution from which it set. It could hold up heavier fragments only after it had set. Before it sets it would not catch heavy fragments, for they would sink. After it sets they would not get into the jelly. Thus one who would explain the isolated heavy fragments enclosed in veins by invoking the colloidal state of the cement encounters the difficulties, mentioned above, which apply to magmas. The difficulties are even greater, since the magma cools slowly and may be viscous through a long period, while the gels set more rapidly and, in the geologic sense instantly, and few fragments falling through a setting colloid would be caught and isolated in the latter.

On the other hand, the fine banding of the cementing material in layers around the fragments, which is so conspicuously shown in certain "ring ores," might be developed in a colloid as Liesegang rings. The Liesegang rings, however, could not completely surround the fragments unless the latter were first surrounded by the colloid, which must catch the fragment the instant of its fall.

FRAGMENTS IN VEINS

Normal metalliferous quartz veins commonly carry numerous fragments of the wall rock and of earlier ore. Some of these are isolated with respect to each other and are supported only by the cementing vein matter. Many of them probably touch other fragments or at one or more points they touch the vein wall, and certain investigators hold that their isolation is more apparent than real. I have been interested in these fragments for many years and have dug into many veins that contain them. I believe that a great many of them are actually isolated.

Six hypotheses may be stated to account for the isolated fragments in veins:

1. They may be floated in heavy solutions or magmas.
2. They may be carried in viscous solutions or gels.
3. They may be dropped from the walls into partly filled veins.
4. They may be formed by reopening of veins along fissures that cross the vein and brecciate the wall rock, and by subsequent deposition of vein matter.
5. They may be residuary or unreplaced portions of the wall rocks of replacement veins.

6. They may be raised into the vein by crystallization of material below the fragments which lifts the fragments above their original points of contact with the walls.

The first two hypotheses have already been treated under the subjects of magmas and colloids. The third has been fully discussed by Lindgren.⁷ The fourth is discussed below.

REOPENING OF VEINS

The formation of mineral veins is often a complicated process. The vein material deposited at an early stage of metallization is commonly fractured or brecciated and the fractures are filled with ore and gangue of a later generation and of different character. This reopening of veins often takes place in the normal course of vein formation when the geologic setting of the vein indicates only one period of metallization. Thus, at Philipsburg, Mont., where there is only one period of vein formation, vein matter consisting chiefly of rhodochrosite is crossed by narrow veins consisting of sphalerite with some quartz and both phases are crossed by veins composed of quartz and a little sphalerite.⁸ The sphalerite veins appear to have formed without much replacement since the rhodochrosite of the first stage of metallization shows sharp fragments and some of them on opposite sides of the veins would fit, if brought into juxtaposition (Fig. 3). The fragments of rhodochrosite are isolated and surrounded by sphalerite and quartz.

Veins formed at great depths and contact metamorphic deposits also show isolated fragments due to movements during the deposition of the ore. In the Cable mine,⁹ 13 miles northwest of Anaconda, Mont., gold ores are deposited by contact metamorphic processes in a mass of limestone almost surrounded by granite. The limestone is recrystallized and at many places the crystals of calcite are an inch in diameter. In the recrystallized limestone are huge blocks of garnet rock which represent an earlier phase of the replacement of the limestone and also huge blocks of the invading granite detached and completely surrounded by the

⁷ *Op. cit.*

⁸ W. H. Emmons and F. C. Calkins: *Geology and Ore Deposits of the Philipsburg Quadrangle, Mont.* U. S. Geol. Survey *Prof. Paper* 78 (1913) 205.

⁹ W. H. Emmons and F. C. Calkins: *Idem*, 221.

recrystallized limestone. These fragments were not floated in the material that recrystallized to form ore and calcite. They were broken up by faulting and the faults were subsequently almost obliterated by recrystallization of limestone. That is shown by Fig. 4, which is a drawing of the wall of the Lake stope. A sill of granite in the limestone was faulted and the limestone was subsequently recrystallized.

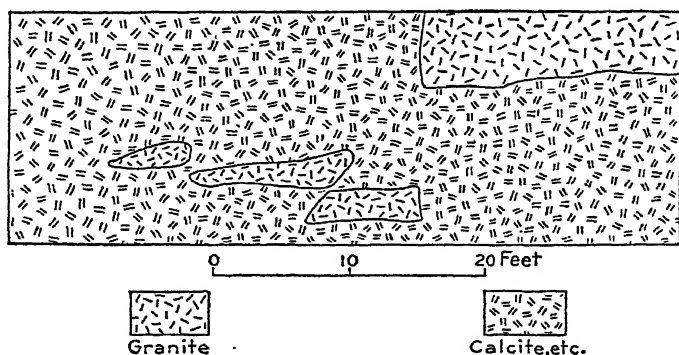


FIG. 4.—VERTICAL SECTION, SOUTH WALL OF LAKE STOPE, CABLE MINE, NEAR ANACONDA, MONT.

FRAGMENTS IN ORE FORMED BY REPLACEMENT

Isolated fragments may appear in ore that is deposited by replacement of the wall rock. Such fragments are characteristic of deposits that have formed by replacement along closely set inclined fractures which strike in three or more directions.



ROUNDED FRAGMENTS
IN REPLACEMENT DEPOSIT

ANGULAR FRAGMENTS
IN REPLACEMENT DEPOSIT

FIG. 5.—DIAGRAMS ILLUSTRATING FRAGMENTS IN REPLACEMENT DEPOSITS.

Isolated fragments in replacement deposits generally have rounded corners because angular corners and edges expose a maximum of surface to the replacing solutions. This condition probably results when the solutions permeate a body of rock and soaking in slowly react with it by chemical interchange, the attack being carried on from all sides. On the other hand, if solutions move more definitely and probably more

rapidly through small fractures, and interchange is effected only along the original fractures at nearly uniform distances from the fractures, the remaining isolated unreplaced fragments will show corners and edges that are acute (Fig. 5). Fragments in replacement deposits have recently been discussed by Bateman.¹⁰

CRYSTALLIZATION OF MATERIAL BELOW FRAGMENTS

Fragments are raised into veins by crystallization of material below them, which lifts the fragments above their original contacts. "Ring ore" is formed by this process. Ring ore is composed of vein material and included fragments of rock or of earlier ore around which there are concentric "rings" or shells of banded ore and gangue minerals. The fragments are clearly isolated and may not touch either wall. They rarely touch each other.

Ring ore forms in vein deposits at all depths. Kato¹¹ has recently described such ore in Dairen copper-tin veins of Akenobe, Japan. Ring ore is found in veins formed at moderate depths, as in rhodochrosite ores of Philipsburg, Mont., and in veins formed near the surface as at Creede, Colo. (Fig. 2). In the Swauk district, Washington,¹² such ore, described by Smith, is found in veins that cut Eocene shales. Such ore is formed by deposition of vein matter around the fragments and completely surrounding them. At some places the vein matter forms successive crusts. At other places it is massive and crusts are entirely lacking. The silver ores of the Alacran mine, Zacualpan, Mexico, which show symmetrically banded crusts, have been described by Spurr.¹³ The crystallization of material cementing such ore has recently been discussed by Taber¹⁴ and by Boydell.¹⁵

In the Hidden Fortune workings of the Homestake mine at Lead, S. D.,¹⁶ one may observe forming today a mineral deposit in which fragments of the wall rock, completely isolated, are being incorporated in the deposit.

¹⁰ A. M. Bateman: Angular Inclusions and Replacement Deposits. *Econ. Geol.* (1924) **19**, 504.

¹¹ T. Kato: Copper-tin Deposits of the Akenobe District, Tajima Province, Japan. *Jnl. College Sci., Tokyo Imp. Univ.* (1919) **43**, 1.

¹² G. O. Smith: Gold Mining in Central Washington. *U. S. Geol. Survey Bull.* 213 (1902) 79.

¹³ J. E. Spurr: Successive Banding Around Rock Fragments in Veins. *Econ. Geol.* (1926) **21**, 520.

¹⁴ S. Taber: Mechanics of Vein Formation. *Trans.* (1919) **61**, 3.

¹⁵ H. C. Boydell: Discussion on Metasomatism and the Linear Force of Growing Crystals. *Econ. Geol.* (1926) **21**, 1.

¹⁶ J. D. Irving: Economic Resources of the Northern Black Hills. *U. S. Geol. Survey Prof. Paper* 26 (1904) 13.

J. O. Hosted and L. B. Wright: Geology of the Homestake Orebodies and the Lead Area of S. Dakota.-I. *Engng. & Min. Jnl.* (1923) **115**, 793.

S. Paige: Geology of the Region around Lead, S. Dak., and Its Bearing on the Homestake Orebody. *U. S. Geol. Survey Bull.* 765 (1924); also N. H. Darton and S.

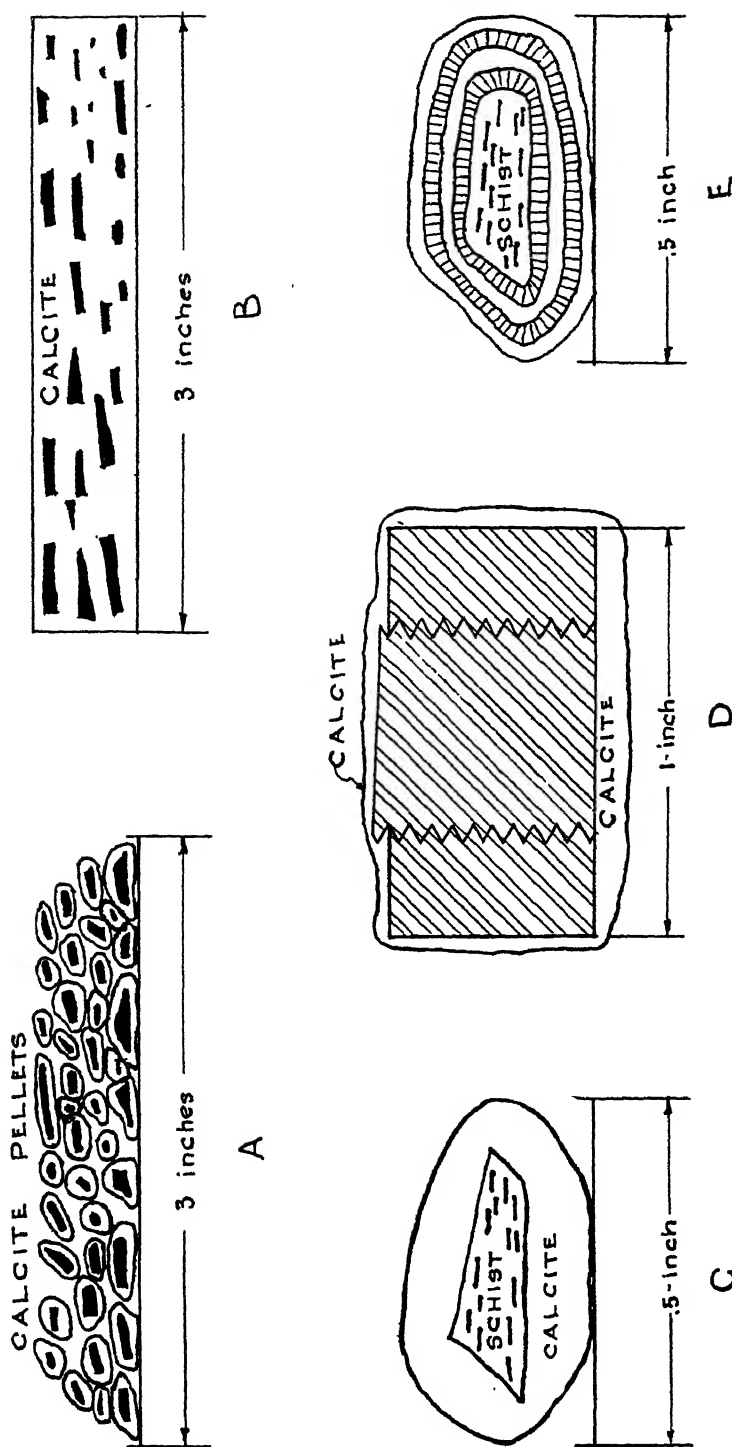


FIG. 6.—FRAGMENTS OF SCHIST FROM HOMESTAKE MINE, LEAD, S. D., COATED WITH LIME CARBONATE OR CALCITE.

A. Pile of pellets now forming in Hidden Fortune adit. The pellets are lightly joined at points of contact. Each pellet has a small core of schist and is completely covered with a thick layer of calcite.

B. A later stage of deposition, showing isolated fragments of schist cemented by calcite.

C. Broken pellet enlarged, showing core of schist.

D. Steel fragments 1 in. across and 0.5 in. thick raised 0.15 in. above floor by deposition of calcite below bolt-end.

E. Broken pellet surrounded by alternating concentric layers of brown and white calcite; "ring ore."

The country rock is Pre-Cambrian schist and the lodes are great auriferous replacements of calcareous members of the schist. The wall rock also is calcareous and waters that now circulate through it are high in lime carbonate. The Hidden Fortune adit is driven southeast in schist. About 920 ft. from the portal, a long crosscut extends northeast. About 250 ft. from the main adit in this crosscut, a small stream of water issues from the rock and flows over the walls and floor. The crosscut is driven in schist and the walls and the floor of the workings are covered with a thick layer of lime carbonate. On the floor there are thousands of small fragments of the schist. Some of these have been cemented solidly together by deposition of lime carbonate. Others have been completely surrounded by crystalline calcite so that they have the appearance of piles of white marbles such as children use for playing games (Fig. 6). Most of these fragments are from 0.25 to 0.5 in. in diameter. At places they are cemented only at points of contact so that they may be easily broken apart (Fig. 6A). A few of the fragments are irregular blocks 4 in. long or more, 3 in. wide and 2 in. thick. These also are thickly coated with lime carbonate. The water is clear and drinkable and is not highly charged with mineral salts except with lime carbonate. Where it issues at the openings, the carbon dioxide escapes and the lime carbonate is deposited. The reaction probably is



Some of the fragments are coated with pure white calcite; others with bands of brown and white calcite alternating. The crystals forming the bands are elongated and lie normal to the surfaces of the fragments. The structure of this material is exactly like that of "ring ore" found in mineral veins. Some of the fragments do not show the alternating bands, but are covered with uniformly white calcite, which under a lens does not appear to be acicular. When these fragments are broken, some are found to have central cores of schist not more than 2 mm. in any dimension, covered uniformly with calcite more than 1 mm. thick. The total amount of calcite greatly exceeds the amount of the schist. Careful scrutiny shows that the lower sides which are at contact with the material on which the covered fragments rest are generally as thickly coated with the calcite as other sides are. Several fragments were more heavily coated on the lower sides at the points of contact than elsewhere. The water is not agitated, the fragments are flat, and they have not rolled over.

The deposit of calcite with enclosed fragments on the floor of the crosscut is 3 in. thick at one place. In the lower part of it, the fragments are cemented into a solid mass several feet long and 2 or 3 ft. wide. The solid mass is 0.5 in. thick and where broken it showed many small fragments which formerly rested on the floor but which had been raised by

the gradual deposition of calcite. Some of the fragments in the upper part of the layer have been raised from the floor about 0.4 in. Under them was white calcite with a few small fragments.

At one place a steel fragment 0.5 in. in diameter and 1 in. across was incorporated in the deposit. The mass of steel was completely covered with calcite and the part of it that rested against the floor was covered with a layer of calcite 0.15 in. thick. Steel has a specific gravity of 7.8, which is about three times that of the schist; yet the steel had been raised 0.15 in. above the floor by the underlying deposit of calcite. A mass of schist nearly cubical in shape, and 3 in. long, was raised 0.1 in. On the lower side of the fragment, and also on the lower side of the steel, rhombic crystals of calcite were forming.

The Hidden Fortune adit was driven about 1900. I visited it and studied the calcite deposit in 1915, and at several intervals two or three years apart after that, the last visit being in 1927. At each visit the water issued into the opening and the lower part of the calcite deposit was submerged at all times. The upper parts of the coated fragments were not always completely submerged. Every fragment seemed to be isolated. I have broken scores of them, which were completely covered with thick layers of calcite.

Deposits of this character are of common occurrence in mines with orebodies or wall rocks of calcareous material. I have noted them in the mines of the St. Joe Lead Co., of Flat River, Mo., and elsewhere.

Force of Growing Crystals

The force of growing crystals¹⁷ has been studied experimentally. It is comparable to the strength of the crystals formed. There is no evidence that large veins are opened by the force, though Becker states that this is possible. Leith and Mead note that sand grains, cemented to form quartzite,¹⁸ show under the microscope surprisingly few contacts. A large proportion of the original quartz grains are not in actual contact but are completely separated by the cement. They suggest the possibility of fragmental particles having been forced apart during cementation of the quartz. The processes that isolate minute grains of quartz during cementation operate also on a large scale to isolate larger fragments.

I was surprised to find that deposition is fully as rapid below the fragments where they rest on the floor as on the sides where no pressure

¹⁷ G. F. Becker and A. L. Day: The Linear Force of Growing Crystals. *Proc. Wash. Acad. Sci.* (1905) 7, 283.

S. Taber: *Op. cit.*

C. K. Leith and W. Mead: *Metamorphic Geology*, 129. New York, 1915, Henry Holt & Co.

¹⁸ R. D. Irving and C. R. Van Hise: Enlargement of Quartz Fragments and Genesis of Quartzites. *U. S. Geol. Survey Bull.* 8 (1884).

is exerted. Experiments show that under some conditions crystals will be dissolved at places of maximum pressure and that the material will be deposited at places where there is less pressure. This is not the case in the Hidden Fortune mine. Where parts of the fragments were above water part of the time, the deposition was greater at the bases than at the tops of the fragments. The small fragments that were completely submerged, apparently at all times, were each covered completely with layers of nearly uniform thickness, although the layers covering certain fragments were thicker than layers covering others. On nearly vertical walls of the workings, minute calcite veinlets are forming in cracks in the schists. Some of these appear to be prying off bits of schist which later might be incorporated in calcite that is forming on the nearly vertical walls, although no completely isolated fragments were noted on the vertical walls. It is possible that a fragment might be separated from a vertical wall and remain essentially in place while it is becoming incorporated in and isolated by the surrounding cement. Such a fragment could remain attached while being separated, since the fragments that are now being raised above the floor are attached and must be struck gently to be dislodged.

LOCALIZATION OF FRAGMENTS IN VEINS

In certain districts the fragments in veins are known to be of distinctly local origin. That is opposed to the theory that they were carried along by heavy magmas. This is shown at Creede, Colo., in the California gold belt and elsewhere.

The Grass Valley district in Nevada County, Calif., is on the west side of the Sierra Nevada batholith. The deposits¹⁹ are gold quartz veins in and near a small domelike mass of granodiorite, which is one of many satellites that lie west of the main intrusive and probably are connected with the intrusive in depth. The chief veins strike northward parallel to the elongation of the intrusive.

The roof of the Grass Valley intrusive consists of diabase and porphyrite of the Calaveras group, and the veins lie in the intrusive and in the invaded rocks. The veins were formed in part by filling openings but according to Howe the deposition was accomplished mainly by replacement of the walls.

The veins carry many fragments of the wall rocks. These are commonly angular and elongate. They lie in all positions often with haphazard orientation. They do not show the alignment like that of elongated masses which are caught up in a molten igneous mass, and they appear not to have moved far from their parent bodies. This feature is

¹⁹ W. Lindgren: *17th Ann. Rept.*, U. S. Geol. Survey (1896) Pt. 2.

E. Howe: *The Gold Ores of Grass Valley, Calif. Econ. Geol.* (1924) 19, 595.

shown very clearly in the North Star vein where it crosses from granodiorite into the diabase-porphry roof above the intrusive. In granodiorite the fragments in the vein are granodiorite and in diabase porphry the fragments are diabase porphry, and granodiorite fragments are wanting.

ORE SHOOTS

It is difficult to reconcile the distribution of primary ore shoots with the hypothesis that ore veins are solidified heavy solutions. It is a theory commonly accepted by prospectors that the intersections of veins are richer than the other portions. This theory has been tested by Roland Blanchard, who has kindly offered me the results of his studies in advance of publication. Of 137 examples of vein systems in which both branches of veins showed commercial ore, 74.45 per cent. showed the better ore at junctions; 11.68 per cent. showed little difference and 13.87 per cent. showed poorer ore at junctions. In a system of connecting dikes formed simultaneously, the dike intersections are generally similar to other parts of the dikes. The materials of a connected vein system are generally much less homogeneous and the causes for the variations commonly are apparent.

VEINS OF CREEDE, COLORADO

In 1923, Spurr²⁰ wrote a paper reviewing a bulletin on the geology and ore deposits of Creede, Colo., by Mr. Larsen and myself.²¹ In this he states clearly and briefly his views regarding the method of vein deposition. He noted that ore veins are generally believed to be deposited by hot waters and says, "I have examined the evidence for this belief and have failed to find the needed proof. I find evidence . . . that many veins have been suddenly filled by injection of a heavy and highly concentrated solution containing metals . . . that this injection penetrating a rift in the rock has distended it . . . producing the fissure . . . and has subsequently frozen to form the vein material." He concludes that the Amethyst vein at Creede was formed by the solidification of a heavy magma, basing his argument chiefly on the presence of angular inclusions of the country rock.

He reproduced two cross-sections of veins from the paper and argues that the fragments could not have been broken from the walls because the walls are regular and show no break. He says also that they could not have been moved away from the wall into the vein by the force of crystallization because the veins are massive and not banded. To him the rude parallelism of some of the fragments indicates the direction of

²⁰ J. E. Spurr: The Filling of Fissure Veins. *Engng. & Min. Jnl.* (1923) 116, 329.

²¹ W. H. Emmons and E. S. Larsen: Geology and Ore Deposits of Creede District, Colorado. U. S. Geol. Survey *Bull.* 718 (1923).

the predominant pressure affecting the heavy solutions at the time of solidification.

I am not convinced of the validity of Spurr's arguments and his conclusion is opposed to evidence which is presented in the U. S. Geol. Survey *Bulletin* reviewed. The walls of the veins are at places regular, yet that does not show that fragments have not been broken from them. As stated in the *Bulletin*, strong fracture planes, generally slickensided, follow one or both walls of the Amethyst vein and the sulfide ore itself is slickensided and at places polished smooth as glass.²² The walls shown in the illustration in the *Bulletin* are not the walls that enclosed that part of the vein when the ores were formed.

The fragments could easily have been moved into the vein by deposition of quartz around them. That quartz has been so deposited is shown by Fig. 10 of the *Bulletin*, which is reproduced as Fig. 2 in this paper.

Parallel fragments are shown at some places in the Amethyst vein, but at other places the longer axes of fragments are normal to each other. Parallelism is shown where isolated fragments are being cemented today in the Hidden Fortune mine by deposition from a solution that is not more than one-seventh as heavy as some of the fragments—by a solution that is in no sense an ore magma, but is in fact a very good drinking water (Fig. 6B).

If the Amethyst vein had solidified from a viscous magma, which in moving lined out fragments, there should be flow lines such as occur in viscous magmas. If it had solidified from a fluid magma, included materials lighter than the magma would float and those heavier than the magma would sink. But the vein (Fig. 1) shows light rhyolite fragments near the foot wall actually below fragments of much heavier galena ore. This figure proves, to me at least, that the Amethyst vein could not have been deposited by a heavy solution.

²² U. S. Geol. Survey *Bull.* 718, 128, Fig. 7.

Aerial Photography as an Aid in Geological Studies

BY GERARD H. MATTHES,* NEW YORK, N. Y.

(Boston Meeting, August, 1923)

ONLY in recent years has any practical headway been made in the application of aerial photography to geological problems, and up to the present time its principal value to the geologist and mining engineer has been in "areal geology;" *i. e.*, the study of formations as revealed at the surface. The object of this paper is to set forth briefly the principal advantages that may be derived from aerial photographic data in geological work and to serve in a measure as a guide to those who may be contemplating its use.

Geophysical prospecting has for object the locating of mineral under the surface. Inasmuch as the geologist's chief complaint in life is a dual one—inability to see under the surface and inability to visualize comprehensively what he does see on top of the ground—geophysical prospecting coupled with aerial photography should do much to lessen his burdens. To what extent aerial photography can be made a useful adjunct to geophysical prospecting remains to be proved, but the inference drawn here is that it holds many alluring possibilities.

To avoid disappointments, it should be stated at the outset that to the geologist or mining engineer an aerial photograph or mosaic is only a tool, the efficient use of which he must set out to master. Upon his skill and ingenuity in applying it to particular problems will depend largely the value of the results that he will obtain. Under the most favorable conditions an aerial mosaic, or even a few loose photographs, because of the exceptionally comprehensive oversight which they afford of field data, such as dips and strike observations at isolated outcrops, local indications of faulting, mineral and fossil finds, greatly facilitate study and correlation, save much tedious foot work, and may repay their cost many times. Even so, types of country may be found where the most skillful use of the best of pictures will produce results so meager as to render the cost of aerial photography entirely unwarranted.

Between these two extremes will be found endless variations and possibilities. Hence no set rules can be promulgated. The safest procedure is to treat each case as a separate problem and to carefully weigh the relative advantages and shortcomings of aerial photography before embarking on its use.

* Consulting Engineer.

INTERPRETATION OF PHOTOGRAPHIC DETAIL

Familiarity with "reading" ordinary maps is no qualification for interpreting the features of the earth's surface as seen in vertical photographs,¹ since man, being a land animal, is not accustomed to viewing objects from the air. Even to experienced geologists physiographic and geological features possess many novel aspects when so viewed for the first time.² Figs. 1 and 2 are reproductions of vertical photographs. The geologist, in this respect, is placed in much the same position as the forester, who, however expert he may be at identifying tree species from the ground, has this to learn all over again when viewing them from above, as he does in an aerial mosaic of a forest. It is therefore indispensable to make a first-hand study of the terrain, mosaic in hand, and thus correlate the new aspect of things with the old. This is recommended even to the experienced whenever an investigation involving new or unusual features is being undertaken. Besides, the study of photographs never can take the place of field examination.

Photography from high altitudes requires the use of color screens or ray filters, to eliminate the effects of atmospheric haze. This results in modified color values, greens appearing dark while bare soil, whether seen in a mere cow-trail or in a freshly plowed field, assumes very light shades bordering on white. This is true even of yellow and red clay soils when dry and of the outcrops of many varieties of rocks which normally would photograph dark. Bodies of deep clear water show up black or very dark except where they happen to reflect the sun, while muddy waters reflect diffused light and appear in various shades of gray, depending on the degrees of their turbidity. Vegetation assumes such widely differing aspects according to locality and season as to defy description here. The subject is not without interest to the geologist, however, as vegetation under certain conditions serves as an index to soil conditions and may indicate the presence of mineral.

Each type of country possesses photographic characteristics of its own and these affect more specifically the clearness with which drainage lines may be followed out. Unexpected extremes are met with in this connection. Thus, the plains of Texas afford the most extraordinarily complete exhibits of water courses—dry for the most part—while in some of the more salubrious sections of the United States vegetation may either mask or accentuate drainage lines. In heavily timbered regions the almost unbroken crown cover of the trees usually hides from view

¹ The term "vertical photographs" applies to exposures made with the camera pointed downward vertically or nearly so.

² A useful introduction to this phase of the subject will be found in *The Face of the Earth as Seen from the Air*, by Willis T. Lee. *Spec. Pub. No. 4*, Amer. Geog. Soc. (1922).



Courtesy of Fairchild Aerial Surveys, Inc.

FIG. 1.—AN AREA IN THE SIERRAS PHOTOGRAPHED VERTICALLY, REVEALING A FAIRLY COMPLETE SEQUENCE OF FORMATIONS.

the smaller water courses. This may be overcome through stereoscopic examination, which brings out the relief and indicates small valleys, gulches, and ravines.

As regards giving expression to topographic relief, aerial mosaics have decided limitations. In regions of bold relief the lights and shades are excellent and bring out the topographic features unmistakably, but mountain tops will frequently be out of position as well as out of scale by appreciable amounts. Gently rolling and plains country, on the other hand, afford no clue to relief, and the latter must be worked out by reference to the drainage. This is not a serious shortcoming, as the stereoscope, with its exaggerated vertical effects, more than makes up for it. Indeed, aerial photographs that are deficient in showing the topography, as a rule are the better "map pictures," in the sense that they show objects in almost true position.

TECHNICAL SIDE OF AERIAL SURVEYING

It has been assumed, for the purposes of this paper, that the reader is familiar with the fundamentals of aerial surveying. The geometrical relations are quite simple, and so far as the needs of the geologist are concerned may be quickly mastered,³ hence will not be described here.

Methods of Aerial Surveying

It has become standard practice to make exposures from airplanes at high altitudes—anywhere from one to three miles above the ground, depending on the requirements of the survey to be made—and to time the exposures so as to secure an abundance of overlap between successive pictures. Although modern aerial cameras have reached a high state of perfection and their performance is truly remarkable, as yet no successful device is at hand for keeping the collimation axis continuously vertical. It still is necessary for the "observer," as he is called in this work, to plumb the camera by hand with the aid of a spirit level prior to each exposure. This, without doubt, is the crudest operation in the entire program of aerial photography and would scarcely be expected to produce acceptable results were it not for cooperation on the part of the pilot by "leveling up" his plane upon signal from the observer when the time for making an exposure draws nigh. Even so, with the best of trying and of team work between observer and pilot, tilting of the camera will occur at the crucial moment.

Tilt, that is the deflection angle between the line of collimation and the plumb line, rarely amounts to as much as 3° , being usually less than 1°

³ See text on Aerial Mapping in Breed and Hosmer's *Higher Surveying*, 2, Chap. 12. New York, 1926. John Wiley & Sons, Inc.

and in high-grade work less than 0.5° , which is commonly considered vertical for all practical purposes. A so-called tilted photograph is one having an obliquity of 2° or more, and requires special treatment before it can be made useful, failing which it may have to be discarded. Tilt, at present, is the chief obstacle in aerial surveying, mainly because its amount and direction are not capable of being determined quickly or exactly except by resorting to ground survey checks. One practical aspect of the tilt situation is that an airplane must be kept on a straight course while vertical photography is in progress. Despite the ever-recurring demand that is being made for it, it remains to this day impracticable to photograph a river valley vertically while following the sinuosities of the river with the airplane.

Areas to be mapped from the air, no matter what their shape or size may be, must be covered by a succession of straight strips of overlapping pictures, generous overlapping being allowed between successive strips so as to avoid gaps. The airplane is flown back and forth in straight parallel courses at as closely as possible the same altitude, the exposures being made by hand or by an automatic timing device, while the camera is turned in azimuth so as to compensate for the angle of "crab" of the airplane. Throughout, the greatest skill is required.

Clearly, this is no job for amateurs, nor are any worth-while results to be obtained except with the best of aerial and photographic equipment. Kodaks and ordinary cameras are all but useless for this class of work.

Up to the present time, single-lens cameras have been employed chiefly for securing photographs for use in geological studies, and in general they are best suited for that purpose. Multilens cameras are superior to single-lens cameras as regards covering large areas cheaply, but the resulting photographs are inconvenient to work with in the field, do not make up well into mosaics except of plains country, and are not adapted to mapping regions of bold relief.

The importance of securing generous overlap cannot be overemphasized. For stereoscopic study alone at least 50 per cent. is needed in order to see any desired point in relief. For geometrical reasons in the construction of line maps, as well as for the best results in constructing mosaics, from 50 to 60 per cent. overlap is an absolute necessity. Finally, ample overlap is the main safeguard for insuring continuity and permits the discarding of occasional defective exposures.

Standard film for single-lens cameras is 24 cm. ($9\frac{1}{4}$ in.) wide and usually 75 ft. long, capable of taking about 100 exposures each measuring 18 by 24 cm. (7 by 9 in.). The camera lenses are ground so as to produce images free from noticeable lens distortions. Standard focal lengths are 8, 10, 12 and 20 in. The use of focal-plane and between-the-lens shutters is about equally divided. Special devices hold the heavy film flat and motionless in the focal plane at each instant of exposure. Development



FIG. 2.—A REMARKABLE VERTICAL PHOTOGRAPH OF A PORTION OF THE COLORADO RIVER DELTA, SHOWING A MAZE OF CHANNELS AMID DENSE BRUSH VEGETATION, THE LATTER BEING AN IMPORTANT FACTOR IN THE DISTRIBUTION OF RIVER DEPOSITS.

and drying of such large films requires special facilities, ordinary photographer's dark rooms being entirely inadequate for the purpose.

Expeditions intending to make use of aerial photography in uncivilized regions should plan with the greatest of care if failure and disappointment are to be avoided. This applies especially to aerial surveys in tropical regions, where such seemingly simple matters as providing ample supplies of cool clear running water, ventilation of dark rooms, maintaining landing fields, providing facilities and repair parts for keeping airplanes and cameras in repair, become momentous problems.

Careful consideration should be given to the selection of the scale of the photographs. It is customary to make single-lens exposures at scales ranging from 600 to 1500 ft. to 1 in. Assuming the focal length to be 12 in., a common procedure would be to photograph from an altitude of 9600 ft. above the average elevation of the ground, thus obtaining a scale for negatives of 800 ft. to 1 in. Enlargement to 400 ft. to 1 in. is then entirely feasible, making prints approximately 14 by 18 in. in size, convenient for use on the plane table. To the geologist the main consideration should be to secure a field scale that will enable him to recognize small surface details, so valuable in locating himself without having to resort to pacing distances, and to be so gaged as to bring out those features with which his study is particularly concerned. At 400 ft. to 1 in., practically all that is of direct interest becomes visible to the experienced eye. In some types of country, as for instance in parts of the arid western United States, scales of 800 and even 1200 ft. to 1 in. will be found ample for field purposes. Enlargement to more than two diameters is not recommended except in special cases, as the prints become unwieldy, and in the end not sufficient additional detail is made visible to warrant the extra expense involved. Many will prefer to use the contact prints with the assistance of a small reading glass, and in the long run this will be found to be the simplest as well as the least costly plan.

Forms of Photographic Data

After vertical exposures have been made, there are at least five distinct forms in which the photographic data may be utilized:

1. *Contact prints*, sometimes called original photographs, made by printing in direct contact with the negatives, in the usual way.
2. *Enlargements or reductions* made by projection from the negatives.
3. *Rough mosaics*, which are assemblages of the central portions of contact prints to form rough photographic representations of the earth's surface. This may be done with or without reference to control points on the ground, and results in reconnaissance maps showing a large amount of detail but not to be trusted as regards scale.
4. *Controlled mosaics*, often called photographic maps, which are assemblages of prints that first have been rephotographed to uniform scale

by reference to ample ground control, thus producing photographic representations of the surface to accurate scale and geographically correct. Ground control, whether consisting of triangulation or of traverses, is indispensable for this purpose, and is usually best provided after the exposures have been made.

5. *Hand drawn plats*, or so-called *line maps*, prepared on tracing linen by transferring by hand the more important features portrayed in vertical photographs. The result resembles in many respects the customary traced maps produced by engineers and geologists, but differs from the latter in not being derived from field survey notes. Here, as in the controlled mosaic, it is important to provide ample ground control. Scale distortions and displacements of photographic images may then be readily corrected and an accurately scalable map produced. This method is recommended wherever ability to scale distances is of importance, as in property maps; also for statistical purposes where colors may have to be used extensively. An important advantage of the line map is the ease with which it may be reproduced. It is not recommended to obtain line maps by tracing controlled mosaics, except in regions of low or moderate relief.

OBLIQUE AERIAL PHOTOGRAPHS

Oblique aerial photographs (see Fig. 3) have been found of decided value in the engineering world and are preferred to vertical photographs for certain purposes. This relates to views where perspective is an aid rather than a detriment; or, stating it differently, wherever scalability, as in a map, is of no great moment. Oblique views possess two manifest advantages: they comprise large fields of view and are therefore more economical than vertical exposures; they give a fair idea of topographic relief, a property in which vertical photographs are deficient except as viewed stereoscopically.

There are times in geological investigations when aerial obliques give at once the quickest and cheapest results, as, for instance, in the mineral prospecting done recently in Northern Rhodesia. Large areas were photographed methodically by the oblique method for the purpose of detecting the presence of copper ores as revealed by stunted vegetation. The same method has been tried for locating oil seepages in tropical regions. The possibilities of mineral prospecting by this method are doubtless capable of further development.

STEREOSCOPIC EXAMINATION

The stereoscope enables one to view the face of the earth in relief by placing two overlapping photographs in proper position under its

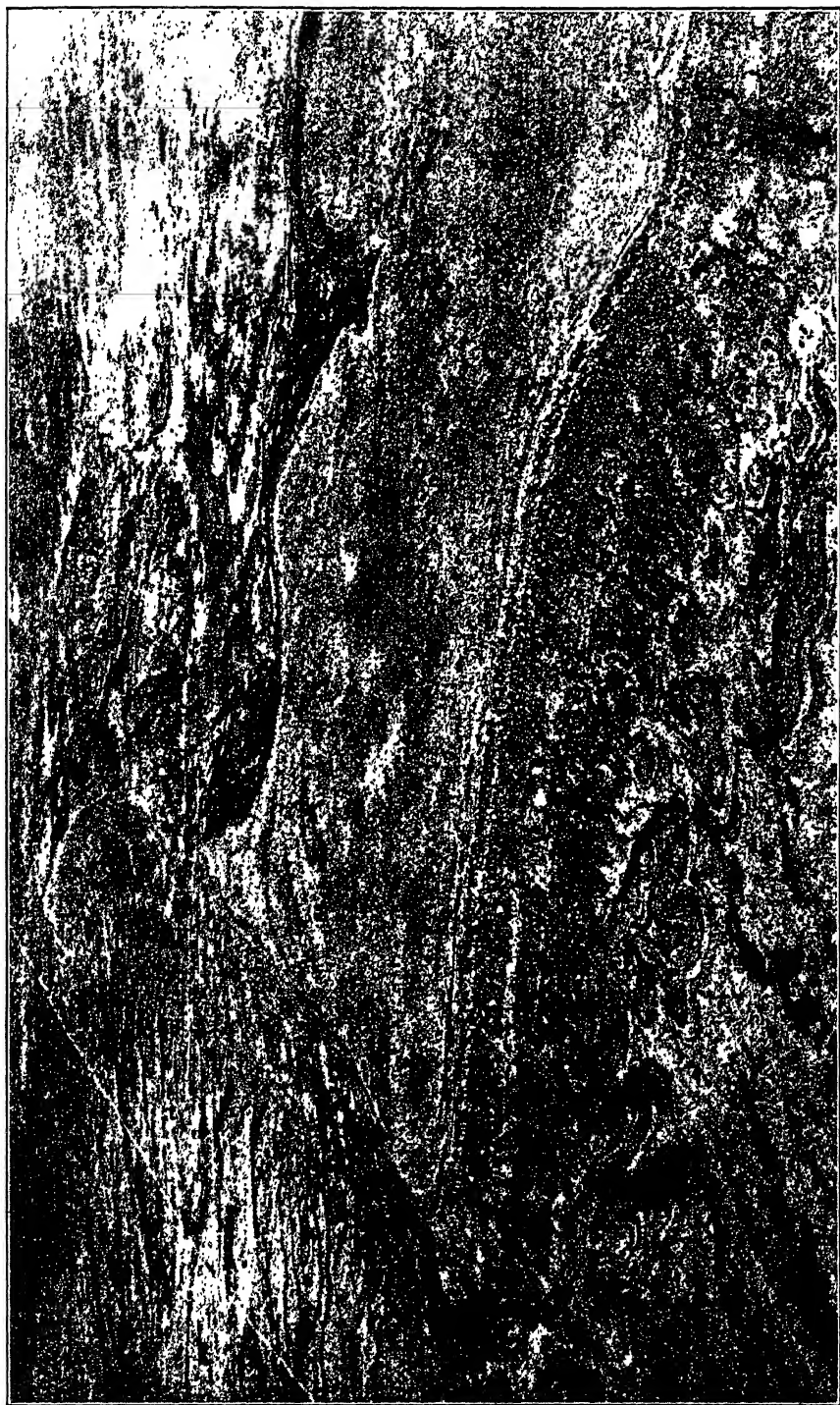


FIG. 3.—OBLIQUE VIEW OF REGION OF SPARSE VEGETATION, SHOWING A LARGE AMOUNT OF GEOLOGICAL DETAIL.
Courtesy of Fairchild Aerial Surveys, Inc.

reflecting mirrors. Only the area duplicated in the pictures will be seen in relief. It is well to note that people who lack vision at one eye, or whose eyes are of quite unequal strength, can get no benefit from the use of the stereoscope. This is because the stereoscopic effect is produced by the optic nerves, not by the stereoscope. The latter merely assists in placing before the eyes pictures taken at points widely separated. This is equivalent to greatly increasing the visual angle and in consequence the relief appears exaggerated. However, this exaggeration of the vertical scale has a pleasing effect on human eyes and assists in bringing out the minor inequalities and slopes of the terrain which otherwise would be scarcely noticeable. While actual differences of elevation cannot be determined by mere stereoscopic examination, an excellent conception of relative differences of elevation may be so gained. To the geologist the stereoscope makes available the same class of information that a relief model does, only better, in the sense that the stereoscope utilizes actual photographs of the terrain. This means bringing into view a wealth of minor detail such as trees, trails, fences, quarries, cliffs, outcrops, highway and railroad embankments and cuts, and a variety of vegetation conditions. Fault lines may become noticeable under the stereoscope which, on a single photograph, may remain too obscure to attract attention. Of great value to the geologist is the convenience afforded of following out fault lines as on a relief model; of noting the slight ridges and valleys produced by certain formations; the continuity of outcrops and key formations; the character of the vegetation as affected by soil conditions; sinks in limestone regions; alluvial lands; form lines produced by outcrops of nearly horizontal strata; the strike of formations as plainly evidenced by outcrops and especially by reefs in the beds of streams where the water is shallow; and, in general, the erosion features of the topography as an index to the physiography of the region.

When aerial photographic work has been well executed and an overlap of not less than 50 per cent. has been obtained between consecutive exposures, the story that may be read out of the pictures under the stereoscope is truly amazing. Nor is this all. Unlike the aviator who can get only very distant views, largely obscured by haze, and who rarely gets opportunity to see any part of the landscape quietly and in detail, the geologist, while comfortably seated in his office or while reclining under a tree in the field, is enabled to study the face of the earth stereoscopically in great detail and as frequently and intently as he may desire.

It now becomes necessary to spoil this good story by stating that aerial photographs on which notes have been made with ink or pencil, as the result of field work, cannot be viewed stereoscopically—the marks either destroy or weaken the stereoscopic vision, at least to most observers. The remedy, of course, is simple enough; it means providing one set of contact prints solely for stereoscopic purposes.



Fairchild Photograph.—Courtesy of The Military Engineer.

FIG. 4.—AN EXAMPLE OF THE HIGHEST TYPE OF CONTROLLED MOSAIC IN WHICH ELEVATION DISPLACEMENTS AND SCALE DISTORTIONS HAVE BEEN CORRECTED THROUGHOUT. THIS MOSAIC WAS MADE FOR AN OIL SURVEY.

USE OF AERIAL PHOTOGRAPHS

For Reconnaissance Work

Controlled mosaics are ideal for reconnaissance work but most often are out of consideration for economic reasons or because they cannot be made without proper ground control. (A controlled mosaic is shown in Fig. 4.) In the absence of control it is necessary to get along with rough mosaics. However deficient in scalable qualities a rough mosaic may be, it often will be found extremely useful for this purpose. This is because location is more a matter of identification by reference to objects portrayed in the mosaic rather than by measuring distances and bearings as in the older methods. In ordinary fairly open country the use of the mosaic often dispenses with the need of any pacing or traversing. The low cost of producing a rough mosaic renders it well suited to reconnaissance purposes whenever an extensive area must be covered in order to locate a few spots of high mineral value. Such spots, once located, being of specific interest, may then be mapped in the form of controlled mosaics by first providing the necessary ground control at each one, and perhaps connecting the individual areas by traverse survey. In any event a great deal of cost will thus be saved in ground surveying and in constructing mosaics. If the photographic work has been well done no additional aerial exposures will be required for making up the mosaics. Regions where mineral occurs in zones or belts may be treated in the same manner.

When time is of great importance it is advisable to dispense with even a rough mosaic and start field operations as soon as the contact prints become available. Good work can be done with contact prints accompanied by an index sheet showing number and location of each exposure. This method appears to have gained favor among the oil geologists on the Pacific Coast and has decided merits; namely, (1) low cost; (2) availability as soon as developing and printing have been finished thus saving the quite appreciable item of time required for putting together any form of mosaic; (3) greater convenience in handling, the 7 by 9 in. prints being readily carried in a compact leather or canvas case; (4) better adaptability to the vicissitudes of rough work and inclement weather than either form of mosaic, the prints, if damaged, being readily replaced at small cost, while for mosaics the reverse is true; and (5) availability for viewing stereoscopically. In the field this may be done with the aid of a light folding stereoscope. It is to be remembered that neither form of mosaic can be viewed stereoscopically.

In other respects the mode of procedure, when working with contact prints, is the same as in the case of the rough mosaic outlined above, a controlled mosaic being made up for each area of specific interest as the

need therefor arises. The contact prints will be found especially useful in determining the exact areas to be covered by such mosaics.

Having dilated on the advantages of working with contact prints, it seems fair to state the disadvantages of this method: (1) the area covered by a single print is limited, therefore the region can not be viewed comprehensively, as in a mosaic (this may be partly offset by the geologist's general knowledge of the region or by reference to a general map); (2) uncertainties as regards scale, a defect shared also by the rough mosaic; (3) inconvenience in spreading out a large number of contact prints in the office when desiring to assemble data for correlation.

The advantages fairly outweigh the disadvantages. The mere ability to start field operations the instant contact prints are available will frequently be the deciding factor in favor of working with the latter.

For Use on the Plane Table

Vertical aerial photographs used singly are admirably adapted for plotting geological data and contour lines by the plane-table method. Where use is made of aneroid or hand level for determining elevation differences, the small traverse type of plane table and a sight alidade are recommended. Where accurate contouring is of importance, it is better to use the standard 18 by 24-in. plane table and a telescopic alidade.

Except in timbered country, it is rarely necessary to resort to plane-table traversing. In reasonably open country it is customary to "set up" at any convenient spot that is easily recognized on the ground and in the picture, and orientation is then effected by sighting at distant objects appearing within the scope of the photograph. Such objects should be selected with a reasonable degree of care and it is important that they be at elevations not materially different from that of the point occupied. The only exception to this rule is when the line of sight happens to pass through the center of the photograph. Any high or low point may then be sighted without the orientation being thrown out by elevation displacement.

The most important advantage of locating geological data by this method is the assistance derived from being able to see objects on the ground portrayed in the photograph. The extent to which this is capable of being carried with the aid of a reading glass and a little practice is astonishing. Images of objects that ordinarily would impress no one as being of any practical interest become important reference points. To locate an outcrop, for instance, by reference to the nearest hillside gully and perhaps a young tree, itself hardly visible in the picture, but still accounted for by its prominent little shadow, become matters of daily routine and save an immense amount of labor.

In contour surveying there is the inestimable advantage of being able to check the position of each stadia shot as rapidly as platted by noting

the position of the rodman with reference to neighboring objects; likewise of being able to check contour location by reference to trees, bushes, boulders, etc., as would come natural to the practiced eye of a topographer.

In country of bold relief differences of elevation may be determined by reading vertical angles and scaling photographic distances. The latter must, in each instance, be corrected for elevation displacement by the radial line method, which may be done graphically in the field with but little labor.

Instead of using single vertical photographs on the plane table, entire mosaics may be utilized as plane-table sheets. The method is not without difficulties where the relief is pronounced, but gives excellent results in regions of low or moderate relief. It is advisable in contour sketching to make the contours fit the photographic detail, thus following out the elevation displacements rather than to try to correct the latter. The resulting inaccuracies are for the most part minor.

For Illustrating Reports

Aerial photographs are effective for illustrating geological reports. Pertinent field and office data may be entered on them with india ink and the photographs are then readily reproduced by rephotographing to any suitable scale. Provided with a north point, bar scale, legend and title, such reproductions are superior to conventional sketches or maps and are especially suited to accompany reports to boards of directors. This is because to the nontechnical man, whether banker or lawyer, such illustrations convey infinitely more than do the old-style maps. Moreover, the nontechnical man takes more naturally to reading aerial photographic detail than he does ordinary maps because the images of natural objects appeal to his imagination.

Oblique photographs add a great deal to any geological report by giving color to the general character of the country under discussion. They convey information of a kind that is never adequately covered by either words or maps.

For Exhibits in Lawsuits

Effective use has been made of aerial photographic exhibits as legal evidence in lawsuits of various kinds. It is entirely conceivable that they may be used with equal success in controversies affecting mineral rights. This aspect is the more interesting because maps of the traditional kind, owing to their conventional signs, confusing contour lines and above all their many blank spaces so unaccountable to the lay mind, have never been popular in court rooms. Aerial photographs and mosaics, on the other hand, though having little legal precedent, from the start

have appealed strongly to both court and jurymen. To quote one California attorney who has used them successfully:

"There is no reason why a photograph of property taken from the air could not be used in evidence in any case that it is pertinent to, providing the proper foundation for its introduction into evidence is laid. By foundation, I mean to show that the aerial survey or photograph is an accurate exposition of the matter of which the survey is made."

Determination of Property Boundaries

Property boundaries usually show up well in vertical photographs if marked on the ground by fences. In many localities a controlled mosaic is tantamount to a property map because it shows in detail the extent and lay of each farm or piece of property, including the buildings and other improvements thereon. This is valuable for securing options or for right-of-way work. However, for property transfers calling for legal descriptions by metes and bounds, aerial photographs will not answer, and the necessary distances and bearings will have to be secured by survey in the usual way. Aerial photographs cannot be made to take the place of that class of surveying.

ADVANTAGES OF CONTRACTING WITH RELIABLE FIRMS

It is no longer advisable for engineers or geologists to undertake aerial photographic work, for the simple reason that it is too difficult to learn to do well. The fact is, aerial surveying has developed into a highly specialized business that calls for costly equipment and specially trained personnel in three distinct vocations, viz.: aviation, photography, and construction of mosaics. Few men are able to qualify in more than two of these vocations, and it takes years of experience to become expert at any one. For the best results and least delay, it is advisable to have aerial photographic work done by specialists and by contract. The same is true of the making of mosaics. At present only commercial firms of long standing can handle mosaic work acceptably. Even the Government bureaus, which do not specialize in that class of work and maintain no specially trained personnel for it, find it difficult to turn out a good product.

As mentioned before, for many geological purposes mere contact prints taken with the proper overlap will suffice. Surveyors, engineers, and geologists can be taught to work with contact prints or enlargements, in both field and office, and to make acceptably scalable plats from them. The principles involved are simple but some practice is needed to acquire skill.

An important advantage that accrues from contracting with competent commercial agencies lies in their willingness to assume all aviation

risks and to protect their clients against damage claims resulting from accidents.

Moreover, there is considerable risk in engaging aerial surveyors who have not been thoroughly trained. Bitter experience has taught that a great deal more is required than mere ability to fly an airplane or take photographs. Inadequate organization and equipment or lack of proper financial backing has spelled ruin to numerous aerial survey companies in the United States, has resulted in contracts being left unfinished, and in serious losses of time to the clients.

SECRECY OF AERIAL SURVEYS

The value of aerial surveys in avoiding publicity has been thoroughly demonstrated by the quite general use that is being made of them by the electric power companies in prospecting for rights of way for important transmission lines. Such surveys are usually made from altitudes sufficiently high above the ground to avoid attracting any attention, and without any ground survey control. In consequence property owners are easy to deal with and important rights of way are acquired at greatly reduced cost. The resulting mosaics, although made without control, except in cases where survey points from earlier surveys happen to be available, are made to as uniform a scale as practicable and satisfactorily serve the purpose of projecting power-line locations. In general, similar methods may be applied to advantage in making mineral surveys from the air where secrecy is of paramount importance.

COST

Contact prints, accompanied by a skeleton index map, usually cost \$8 to \$10 per exposure, with variations introduced by the size and shape of area to be covered and by climatic conditions. To this should be added the expense of bringing the airplane and crew to the scene of operations and for their return journey, allowing \$1 per linear mile flown. Scale is an important factor in affecting costs as shown in Table 1

TABLE 1.—*Number of Single-lens Exposures Required at Various Scales.*

Scale	Exposures per Linear Mile of Flight	Exposures per Square Mile
1 in. to 1200 ft.	1.57	1.54
1 in. to 1000 ft.	1.89	2.21
1 in. to 800 ft.	2.36	3.46

For controlled mosaics the costs as stated above, are increased at the rate of \$10 per exposure. This does not include ground survey control for which no estimates can be given here owing to their extreme variability.

Geology of the Moffat Tunnel, Colorado*

By T. S. LOVERING,† WASHINGTON, D. C.

(New York Meeting, February, 1928)

THE Moffat tunnel passes through the continental divide about 50 miles west of Denver, Colo., on the Denver & Salt Lake R. R.¹ It is 16 ft. wide, 24 ft. high, and 32,383 ft. long, a distance of a little more than 6 miles. The advantages accruing from its construction may be briefly summarized as follows: The tunnel eliminates 27 miles of grade averaging more than 3.5 per cent. and actually running well over 4 per cent. in many places, and changes the steepest grade from 4.5 to 2 per cent. It reduced the peak of the railroad from 11,660 to 9249 ft. which enables one engine to pull four times as much freight as it could before the tunnel was built. If the Denver & Salt Lake R. R. completes its line, this road will shorten the distance between Denver and Salt Lake City by 200 miles and will have no steeper grades than the lines now connecting the two cities.

The proposal to drive the tunnel through the divide dates back about 25 years and D. H. Moffat, who was the leading spirit in the organization of the Denver & Salt Lake R. R. spent much of his time and energy in efforts to raise money for the tunnel that bears his name. He was unsuccessful and died before the work was started. In 1920 an effort was made to obtain state legislation favoring a bond issue to finance three important railroad tunnels in Colorado but the measure was defeated, largely through the efforts of the citizens of the region near Pueblo. A few years later devastating floods caused the Pueblo voters to seek relief from the state and at this time they withdrew their objection to the tunnel bond issue. As a result, a \$6,720,000 bond was issued to finance the Moffat tunnel. Construction bids were so high that the work was finally allotted on a scheme analogous to a cost-plus basis. The contract was awarded to Hitchcock and Tinckler, who are paid a fee of \$140,000 for driving the tunnel. All bills are paid by the Moffat Tunnel Commission. The contract was let Sept. 19, 1923, and Hitchcock and Tinckler agreed to complete the work by July 19, 1927. In the event of finishing before this date they were to be given a bonus of \$1000 a day and

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¹A well illustrated description of the tunnel, its history, the methods of construction employed, and the organization of the personnel can be found in a series of articles by R. G. Skerrett: *Driving a Tunnel through the Continental Divide. Compressed Air Magazine* (1925) 30, 1133, 1165, 1205, 1237.

if completion was delayed to a later date they agreed to pay the Moffat Tunnel Commission \$1000 a day for liquidated damages. Excavation was completed Dec. 10 and trains will probably pass through the tunnel before the end of February.

Before the tunnel was driven the ground was carefully examined by the State Geologist and several engineers. Based on these preliminary surveys the total cost of the undertaking was estimated at \$5,250,000. The actual cost has been approximately four times this figure. The original estimate was based on the assumption that the tunnel would be driven entirely in strong solid rock. In reality about two miles of the tunnel is in weak material that requires substantial support. Oregon fir, gunite, steel sets and reinforced concrete have all been employed to hold the heavy ground and in the worst zone more than 2 tons of steel per running foot of tunnel are employed to reinforce a high-alumina, quick-set concrete jacket which is 32 in. thick in the bottom, 29 in. in the roof, and from 30 to 45 in. thick in the sides.

A pilot bore called the water tunnel was driven 75 ft. south of the railroad tunnel. The 9 by 9 ft. heading of this pilot opening was driven in advance of the main tunnel and served to explore the ground and prepare the engineers for their difficulties. Crosscuts between the two tunnels at intervals of about 1500 ft. allowed material to be handled with great efficiency and greatly expedited the construction of the railroad tunnel. Nearly all the bad ground was encountered near the west portal of the tunnel, the most troublesome material was gradually entered about 12,000 ft. east of the west portal and extended another 1000 ft. to the east.

In this zone the ground swelled continuously and almost irresistibly. In the water tunnel, 12 by 18-in. timbers of Oregon fir were broken like matchsticks. The timber used in 100 ft. of this tunnel—not including any replacements—ran as high as 89,000 foot-board measure of Oregon fir. Much of the timber failed in a few days when the ground was first opened up but is holding much better at present. Similar but greater difficulties were encountered in the main tunnel and, of course, increased driving and construction costs enormously.

Failure in the water tunnel was frequently reflected in the railroad tunnel 75 ft. to the north by a strong southward drag. For this reason the engineers determined to place a concrete jacket in the water tunnel in the zones of heaviest ground.

As the concreting of the water tunnel would permanently hide the wall rock, it seemed advisable to make a geological study of this 6-mile cross-section of the Rocky Mountains before important parts of the geology were sealed off. The purpose of the study was threefold: (1) to record the details of the geology and structure; (2) to determine the cause of the heavy ground; (3) to note how much of the structure revealed in the tunnel is reflected on the surface.

GEOLOGY OF THE DIVIDE

The Moffat tunnel is almost wholly in Pre-Cambrian rocks but a thin cover of Quaternary glacial debris and Recent slope wash covers the ancient bed rock at either end of the tunnel. The chief rocks encountered were injection gneiss, pegmatite, quartz monzonite, granite, and schist. Like many Pre-Cambrian regions, the general structure is simple if viewed in a broad way but is exceedingly complex when studied in detail. For a general picture, it is sufficient to realize that the tunnel passes through two anticlines, separated by a strong fault zone known as the Ranch Creek fault, which occurs about 2 miles east of the west portal. Schist, with minor amounts of pegmatite and injection gneiss, is the chief rock west of the Ranch Creek fault and quartz monzonite and injection gneiss are the chief rock east of the fault. In addition to the Ranch creek fault there are many subordinate fracture planes having gouge seams 1 to 12 in. thick. They are more frequent in the schist area than in the eastern part of the tunnel. They are not evenly spaced but the average distance between them is 500 ft. on the western side of the Ranch Creek fault and 1000 ft. on the eastern side. Nearly all the fracturing of the quartz monzonite and injection gneiss occurs along cross breaking faults, but in the schist there has been a large amount of movement along bedding planes as well.

Along the Ranch Creek fault the maximum movement occurred in a zone about 200 ft. wide, where greatly altered masses of schist, granite, gneiss, and quartz monzonite are broken and intermixed in a network of thick gouge seams. To the east, within the comparatively short distance of 250 ft., this material grades through bleached and shattered granite into strong fresh-appearing granite and quartz monzonite. From this point to the east portal, practically all of the tunnel is untimbered. For 600 ft. to the west of the zone of maximum movement, the ground improves very gradually. Throughout this belt the schist is cut by many strong cross faults and in addition has undergone a tremendous amount of interlaminal movement. The western edge of the fault zone is fairly well defined and within 50 ft. the character of the rock changes from weak swelling ground cut by many gouge seams to fairly strong, massive quartz-sillimanite schist.

The rock in the Ranch Creek fault zone is moist but no water-bearing fissures occurred in this part of the tunnel. The only large flow was tapped in the eastern half of the tunnel about 8000 ft. from the east portal. A narrow fissured zone dipping east at 45° yielded 1800 gal. per min. when it was first tapped. This flow soon decreased and dropped to 150 gal. per min. within a month. This fissure is directly beneath Crater Lake and the level of the lake fell several feet as a result of this subsurface drainage. A few other water-bearing fissures were encountered but

water has never been a serious problem for more than a few days at a time.

HEAVY GROUND

Any rock that requires sills, posts, caps, side and top lagging, is classed as heavy ground. In the Moffat tunnel heavy ground is due in part to the character of the schist and the gouge-filled minor faults, but especially to the Ranch Creek fault zone. Much of the distance through schist between the west portal and the Ranch Creek fault is in heavy ground, although there are moderate distances through strong ground that require little or no timber. In general, the timber replacements are moderate through this section of the tunnel and anyone familiar with the mines in the near-by schist areas would find nothing unexpected in the behavior of the rock. The inherent weakness of the schist has been accentuated by crumpling and differential movement in the softer layers and substantial support is needed when passing through this type of rock.

The minor faults that carry gouge seams more than 8 in. thick commonly have walls of altered rock, and strong timbering is usually required in passing through them. At some of the strongest of these faults the first timbers failed within a few days. The forces causing the failure acted like hydrostatic pressure; in the first sets used the mud sills were bent up into the tunnel and broken, and when octagonal sets of nearly uniform strength were substituted, failure occurred in an apparently haphazard way, successive sets breaking indiscriminately at top, bottom, or sides. The use of 12 by 18-in. Oregon fir in octagonal or inverted arch sets largely overcame the difficulties presented by these strong minor faults. The wet clay gouge and the moist kaolinized walls of the faults were little stronger than a stiff mud and acted like a viscous fluid under the rock pressures existing in the tunnel.

Although the section west of the Ranch Creek fault differs little in general character from the schist bodies encountered in mines of the Georgetown or Central City quadrangles to the east, the Ranch Creek fault zone would probably be matched in very few, if any, of the underground workings in this part of the state. It is impressively bad. For a distance of nearly 1000 ft., the full width of the fault zone, the tunnel at no place is out of soft swelling ground. The roof, walls and floor close in steadily at rates varying to a maximum of 3 in. a day. Timbers of Oregon fir 12 by 18 in. are cracked and splintered before the steady advance of the swelling ground. On an average, the standard sets of native timber lasted less than three days. The heavy inverted arch and octagonal sets lasted from two to three weeks when first put in but at the present time are being replaced only once in two to four months.

The walls are deceptive in appearance throughout this belt of swelling ground. In the western part of it the schistosity is fairly consistent for hundreds of feet, and gouge seams, though very numerous, are generally narrow. Yet it is almost impossible to find a piece of rock 1 cm. across that does not show slickensides when broken. On the eastern side of the fault zone the walls are equally deceptive, very few gouge seams are present and the rock seems to be a bleached but homogeneous granite. It is quite surprising to observe material of this appearance slowly closing up the tunnel without apparently developing any cracks or shear planes. In fact, the behavior of the ground impresses experienced hard-rock miners as so uncanny that they will seldom work in this part of the tunnel for more than a few days.

Samples of this heavy ground present a surprising picture when studied under the microscope. The granite and quartz monzonite grade from fresh unbroken rock into a slightly broken mass cemented by microscopic veinlets of quartz and a potash-bearing clay mineral related to montmorillonite. This broken rock becomes progressively more shattered as the fault zone is approached and close to the worst ground the material that resembles bleached granite really consists of a minutely brecciated mass of quartz and feldspar poorly cemented with chalcedonic quartz, and the potash-bearing clay.

CAUSE OF HEAVY GROUND

In considering the problem of the swelling ground, an effort was made to test all the possibilities that could possibly account for the behavior of the rock. The possibilities are relatively few and can be considered in two groups: the chemical causes and the physical causes. Swelling ground is often ascribed to the indefinite process called "air slacking." The chemically active substances found in the air are oxygen, carbon dioxide and water. No sulfides were found in the swelling ground and none of the minerals observed can take up oxygen from the air. The carbonates in the shattered rock are not appreciable and no test for carbon dioxide can be obtained in rock that had been exposed to the air for several months.

The moisture content of the rock varies with the humidity of the air. About 2.35 per cent. of moisture can be driven out of fresh rock at 110° C. After a short time in an atmosphere of low humidity—about 12 per cent.—the moisture decreased to 0.97 per cent. Material dried at 100° did not swell perceptibly when moistened. Only rough measurements have been made at present but it can be stated definitely that any increase in volume due to absorption of water is less than one part in one thousand.

Pulverized material was placed in a cylinder and compressed by a load of 25,000 lb. per sq. in. A certain amount of elasticity was indicated

when the load was removed from the dry powder, which expanded about 1.5 per cent. This material absorbed about 2.5 per cent. moisture without any change in volume. However, the addition of moisture greatly changed the compressibility and elasticity of the rock powder. When this moist material was loaded it yielded readily and its volume under 25,000 lb. per sq. in. was only 87 per cent. of its volume when dry under the same pressure. When the load was removed, no elastic recovery could be observed. It seems very probable that its decrease in volume reflects the lubricative qualities of the moist clay flakes, which greatly decrease the friction between the angular fragments in the brecciated mass and allow them to slip easily past one another into the position of closest packing.

Thus none of the possible chemical causes of swelling ground, oxidation, carbonation or hydration—seems adequate to explain the effects observed. The swelling, therefore, must be due chiefly to physical causes. The weight of the rock above the tunnel near Ranch Creek is approximately 1200 lb. per sq. in. The experiment last cited indicates that even under much greater loads, the heavy ground does not suffer appreciable elastic compression. This makes it wholly improbable that the tunnel relieved elastic compression and that the swelling ground represents elastic expansion.

Two striking differences exist between the swelling ground and the good ground. The strong rock is relatively solid and is not appreciably altered. On the other hand, the swelling ground is thoroughly shattered and has the potash-bearing montmorillonite clay strongly developed throughout. In these differences must lie the cause of swelling.

The clay mineral in the bad ground was isolated from the granite and the pure mineral was studied. Its analysis is given in Table 1 and its behavior under varying conditions of humidity is shown. Its analysis, optical properties, and X-ray diffraction pattern² show that it is the clay mineral described by Ross as a potash-bearing montmorillonite.³ The large amount of moisture taken up under ideal conditions readily explains the variation of moisture with humidity previously noted in the granite. The clay has the hardness of a sun-dried brick in an atmosphere whose humidity is about 12 per cent., but if this same clay is moistened it becomes a soft, highly plastic, slippery mass, devoid of strength. When a mineral having characteristics such as these becomes an important constituent of a rock, it is a foregone conclusion that the rock will be weak and unsound. When the moist clay forms an important part of a

² Dr. Paul F. Kerr of Columbia University photographed the X-ray diffraction pattern of the clay mineral and finds that it is identical with a high-potash montmorillonite from High Bridge, Ky.

³ C. S. Ross and E. V. Shannon: The Minerals of Bentonite and Related Clays and Their Physical Properties. *Jnl. Am. Ceramic Soc.* (1926) 9, 92.

thoroughly shattered rock the resulting mass will be incapable of withstanding even moderate stresses. It is believed that the moist flakes of clay act as a lubricant along the innumerable free surfaces in the brecciated rock of the fault zone, and allow it to move as a highly viscous mass under the rock pressures which obtain in the tunnel. Swelling ground thus expresses the summing up or integration of minute slippages between extremely small fragments in the shattered granite or schist.

TABLE 1.—*Potash-bearing Montmorillonite from Shattered Granite of Ranch Creek Fault Zone, Moffat Tunnel*¹

Chemical Analysis		Humidity, Per Cent.	Relative Weight of Clay
Mineral	Per Cent.		
SiO ₂	58.27	0.00	1.00
Al ₂ O ₃	20.30	11.7	1.029
Fe ₂ O ₃	3.00	21.7	1.042
FeO.....	0.23	41.3	1.070
MgO.....	1.79	64.7	1.091
CaO.....	0.82	80.5	1.123
Na ₂ O.....	1.91	94.5	1.175
K ₂ O.....	4.96	98.5	1.202
H ₂ O - 110°.....	3.85	100	1.253
H ₂ O + 110°.....	5.04	ignited clay	0.942
Total.....	100.17		

Index of refraction of material analyzed, at room temperature $\alpha = 1.525$, $\gamma = 1.545$, $\gamma - \alpha = 0.20$.

¹ Chemical analyses by J. G. Fairchild, U. S. Geological Survey. Determinations of moisture in relation to humidity, by P. G. Nutting, U. S. Geological Survey.

In the fault zone the rock pressure is transmitted through material capable of yielding as a viscous mass and is therefore essentially all sided or hydrostatic in its nature, causing a uniform inward movement of the bad ground from top, bottom, and sides of the tunnel. As the material dries out the internal friction would become greater and the rate of movement should tend to lessen. However, this factor may be relatively unimportant and if the tunnels were kept open by continued shaving of the walls the swelling should decrease slowly even though the rock remained moist. A highly viscous mass does not transmit pressure rapidly and in the same way any changes in rock pressure would be adjusted slowly in the zone of swelling ground. Thus the removal of a certain amount of material at once relieves the pressures in adjacent rock; material a short distance away is under less stress than before but distant rocks are not affected at all. The stress gradient in all parts of the rock mass will slowly readjust itself to the new conditions. The adjustment will be rapid near the tunnel but probably quite slow at distances of

1000 ft. As time goes on, the rock pressures a short distance from the opening will become less and less important and this change will be reflected in a slower movement of the bad ground. Thus the expedient of maintaining an opening through the swelling ground would gradually reduce the swelling of the heavy ground to unimportant amounts.

RELATION OF SUBSURFACE TO SURFACE GEOLOGY

One of the objects of the geological study of the Moffat tunnel was to determine the manner in which the subsurface geology was reflected at the surface. Unfortunately, there was so much snow on the surface last November that a detailed study could not be made. It is hoped that such a study may be made next summer. However, from the small amount of surface work that was done last fall, two things are fairly certain. First, the Ranch Creek fault is not conspicuous on the surface and it is quite possible that it and the altered granite to the east of it cannot be detected in the glaciated valley where it reaches the surface. The change from schist to granite, therefore, could be reasonably—though mistakenly—interpreted as an intrusive contact. Second, the soft schist which gave the engineers so much trouble west of the Ranch Creek fault is well exposed by cuts along the railroad. In some of these cuts heavy gouge seams are conspicuous. Outcrops of schist are rare along the line of the tunnel west of Ranch Creek but pegmatite exposures are more common. Thus the field facts show (1) that schist is the chief rock west of Ranch Creek, (2) that faults and gouge seams are common in the schist, (3) that the schist is not resistant to weathering and would probably be relatively weak. Knowledge of mining in the near-by schist areas would prepare an engineer in possession of these facts for the conditions actually encountered underground west of the Ranch Creek fault.

SUMMARY

The chief facts brought out by the geological study of the Moffat tunnel are as follows:

1. A fault zone 1000 ft. wide striking a little east of north exists about 2 miles east of the west portal.
2. The chief rocks east of the fault zone are granite, quartz monzonite and injection gneiss, whereas schist is the most important rock found on the western side of the fault.
3. The formations on both sides of the fault show distinct anticlinal structures.
4. The schist was relatively weak and required much timbering but the injection gneiss, granite and quartz monzonite required practically no timbering.

5. The swelling ground in the wide fault zone was not caused by air slacking.

6. The swelling was caused by rock pressures on a thoroughly shattered rock, and movement of small fragments on innumerable surfaces lubricated by a wet clay mineral accounts for the swelling of the rock.

7. The clay mineral that is largely responsible for the swelling is a potash-bearing montmorillonite.

8. The swelling should become less as time goes on if the tunnels are kept open by shaving the walls.

9. The heavy fault zone is not conspicuous at the surface.

10. The character of the soft schist, which required much timbering, can be inferred from an examination of the surface.

DISCUSSION

MEMBER.—Were there any mineralized areas anywhere in the tunnel?

T. S. LOVERING.—Some in the strong schist; these were sporadic and rather local and seemed not to affect the strength of the rock that the tunnel went through. There were some copper seams on many of the fissures that occurred in the eastern part of the tunnel, within the first 2000 ft. Some copper stain was found locally in the heavy ground of the Ranch Creek fault zone.

E. L. WILSON, Cambridge, Mass.—Do you believe that montmorillonite was developed by dynamic metamorphism or that it was due to ground water in fairly recent times?

T. S. LOVERING.—I think there is no doubt that montmorillonite was developed by cold waters going through shattered rock. It occurs chiefly in small flakes replacing the feldspar, and along these veinlets which cross through the shattered rock. It is part of the cement or groundmass which was evidently formed later than the shattering of the rock.

D. WHITE, Washington, D. C.—What is inferred as to the fault zone?

T. S. LOVERING.—The fault zone is rather steep dipping and I believe that the eastern side has dropped down. More than that I can hardly say. My chief reason for believing that the eastern side has dropped down is that the schist is younger than the injection gneiss.

D. WHITE.—You mention horizontal stress in the movement in the big tunnel from south to north.

T. S. LOVERING.—That was in the tunnel itself. There were movements reflected in the water tunnel or in the railroad tunnel, according to which failed first. It was simply a viscous mass; if it was relieved in one place, everything moved that way.

W. LINDGREN, Cambridge, Mass.—What preliminary work was undertaken?

T. S. LOVERING.—The state geologist examined the region and made a report on it and there were some engineers there, also, I believe. Mr. Finch probably knows more about the detailed work done about that time than I do. He was connected with the Moffat Tunnel Commission, if I am not mistaken. There was a preliminary report, which was all right up to the Ranch Creek fault. I have seen only the cross-section, but from the Ranch Creek fault zone to the western portals, it was wrong.

The report had the gneiss dipping toward the west; as a matter of fact, the rock is schist, dipping toward the east. Except in this region of bad ground, the cross-section is excellent.

MEMBER.—Was there a geologist on the ground while the tunnel was in progress, to keep the information up to date?

T. S. LOVERING.—There was no geologist on the ground; and it was rather a surprise, to me at least, that practically all the work was conducted by civil engineers instead of mining engineers.* I think some of the trouble would have been anticipated if there had been men who were a little more familiar with underground conditions in similar rocks near by.

J. W. FINCH, Denver, Colo.—It is true that I was connected with the first commission that was created to study the conditions along the proposed route of the Moffat Tunnel. I was not a member of the commission, but of the Engineering Advisory Board.

The Engineering Board was given the duty of estimating costs. We stated to the Tunnel Commission that the estimate of costs must necessarily be based upon geology. We would take no responsibility for that.

The state geologist went over the ground. He was requested particularly to look for fault zones, and as Mr. Lovering has indicated, the big faults made the trouble. He reported that there was schist at the west end, but that it might not be troublesome; the other rocks looked firm and strong. In defense of the state geologist's report, we must say that the tunnel plan was developed at a time when public enthusiasm was necessary. The tunnel is a public enterprise financed by taxation. Things were being done with a great rush, and he was induced to make a hasty examination. At any rate, the Engineering Board was given data upon which it was considered safe to estimate a moderate amount of timbering and our estimate of total cost was, I believe, \$7,500,000.

We were wrong, we admit, in the first estimate, and we admit that there were other things wrong, particularly in the lack of intensive geological study.

D. WHITE.—Were any temperature curves drawn?

T. S. LOVERING.—No.

H. C. BOYDELL, Toronto, Ont.—Dr. Lovering mentioned that the rock pressure in the swelling ground was actually measured to be about 1200 lb. per sq. in. How did that compare with the calculations?

T. S. LOVERING.—I did not make that clear I am afraid. My calculation of 1200 lb. per sq. in. is based on the distance to the surface and density of the rock assumed as about 2.6. The actual pressure necessary to cause timber failure there runs only about 250 lb. per sq. in., and no measurements were made excepting when the sets were changed. When the timber gave, they put in a stronger set, and when that failed, a still stronger set; from that they went to steel, then to concrete, then reinforced concrete, and then quick-set cement. That was the only measure. I meant that the maximum pressure due to the weight of the rocks would be 1200 lb. per square inch.

Landslide and Flood at Gros Ventre, Wyoming*

BY WILLIAM C. ALDEN,† WASHINGTON, D. C.

(New York Meeting, February, 1928)

A GREAT landslide occurred on June 23, 1925, in the valley of Gros Ventre River, about 35 miles south of Yellowstone National Park (Fig. 1). The relations of the north-easterly dipping rock formations of the slide scarp and of the dam formed by the slide are shown in the diagrammatic cross-section of the valley (Fig. 2). This generalized section is based on unpublished maps and notes by Eliot Blackwelder, in the files of the U. S. Geological Survey.

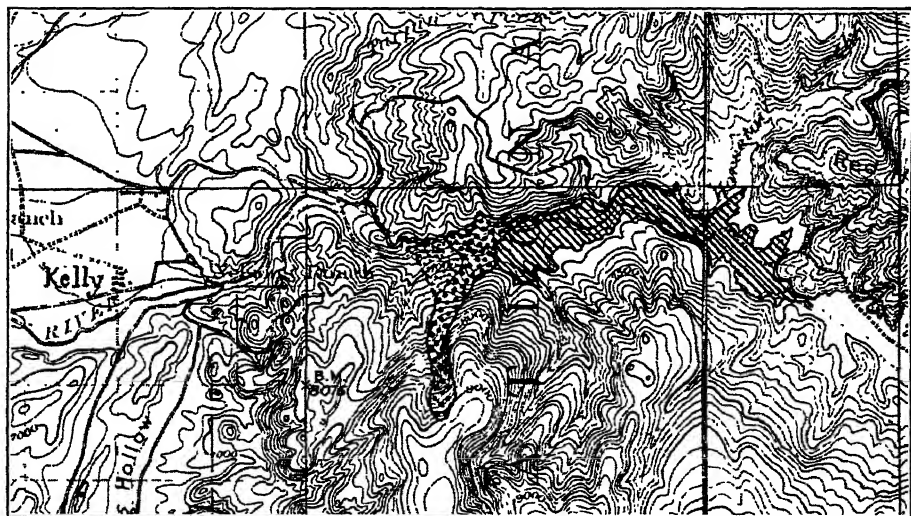


FIG. 1.—MAP SHOWING SCARP AND DAM (LARGE DOTS) AND LAKE (CROSS-LINED) FORMED BY LANDSLIDE BLOCKING GROS VENTRE RIVER.

(Scale $\frac{1}{2}$ in. = 1 mile; contour interval 100 ft. From Grand Teton and Mt. Leidy topographic maps).

Heavy rains and melting snow in the Gros Ventre Mountains had saturated clay layers in the Carboniferous strata, which dip toward the valley at angles of 18° to 21° , consequently an enormous mass of rock at the end of the north spur of Sheep Mountain became loosened and, on the afternoon of June 23, slid suddenly down into the valley. Within a few minutes this mass of debris—estimated as 50,000,000 cu. yd.—carrying

* Published by permission of the Director, U. S. Geological Survey.

† U. S. Geological Survey.

on its surface a dense pine forest, rushed across the valley, piled up about 350 ft. high against the cliffs of red sandstone on the north and partly slumped back and spread so as to form a dam 225 to 250 ft. high above the lower toe of the dump. The path of the slide and the resulting dam are shown in Figs. 1 and 2. The river, which was in flood, was completely blocked and rapidly filled the basin behind the dam, so that within 18 hr. a ranch house standing 60 ft. above the river was floated off its foundation and in about three weeks the lake (shown by the cross-lined area in Fig. 1) reached a depth of about 200 ft. at the back of the dam, an average width of about $\frac{1}{2}$ mile, a length of about 3 miles, and an estimated area of about 11,000 acres. Fig. 3 is a distant view looking up Gros Ventre valley



FIG. 3.—DISTANT VIEW OF SCARP OF LANDSLIDE ON NORTH SPUR OF SHEEP MOUNTAIN. SLIDE DAM AND LAKE NOT IN THIS VIEW.

to the slide scarp. The dark cross line is of trees on top of a great mass which lodged several hundred feet below the top of the scarp. The slide dam and the lake are not seen in this view.

Fig. 4 shows the scarp and the path of the slide as seen from the opposite, or north, side of the valley in September, 1925. The head of the scarp is about $1\frac{1}{2}$ mile south of the river channel and about 2100 ft. above it. Below the top is the line of trees standing on a mass which slid down a few hundred feet and lodged there. At the left, about 1000 ft. above the lake, is a lateral offshoot of the slide which was projected into the dense timber. At the lower right side of the view, is the upper part of an enormous cracked and broken mass, with forest trees tilted helter skelter, which started to move but lodged. If the river continues cutting at the foot of this mass, renewed sliding may result.

Soon after the slide, water began seeping through the dam. This took care of the inflow, which decreased as the summer came on. On Aug. 18,

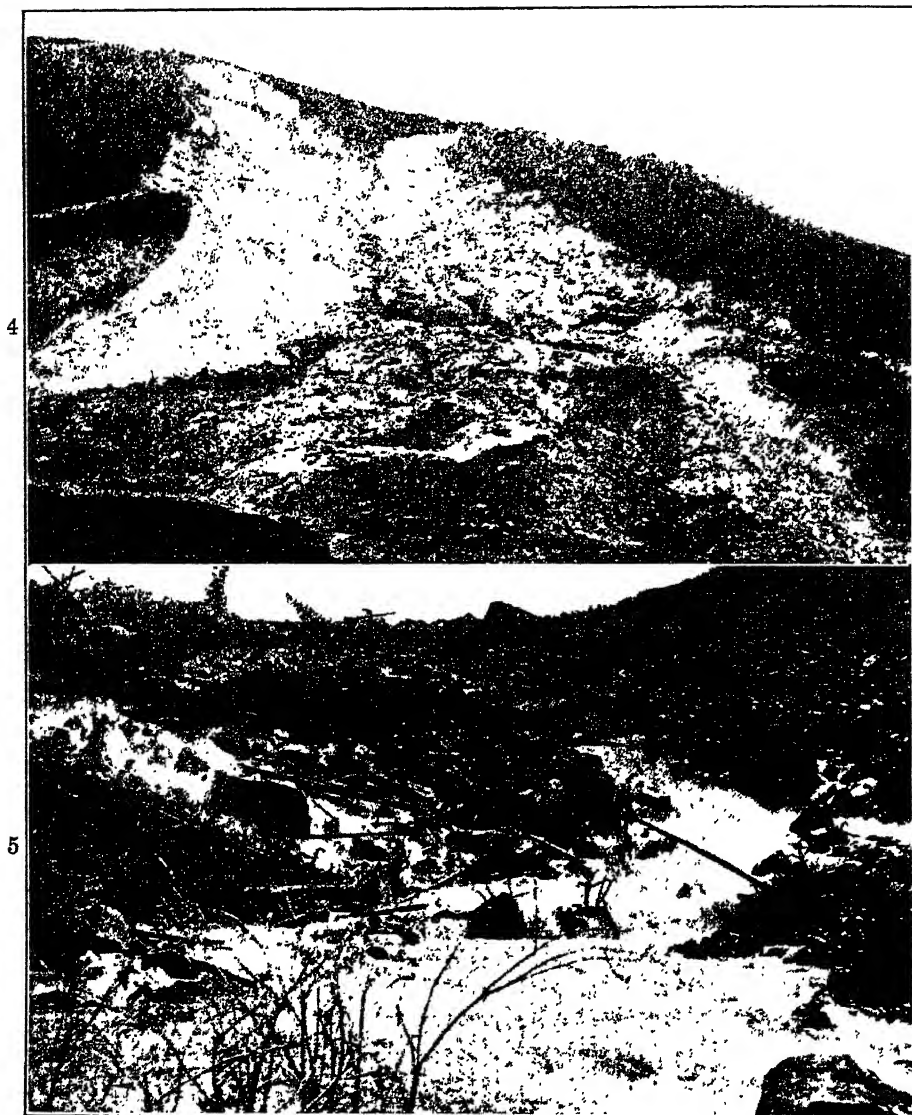


FIG. 4.—VIEW UP SLIDE SCARP FROM NORTH SIDE OF GROS VENTRE VALLEY. HEAD 2000 FT. ABOVE LAKE. LINE OF TREES ON A MASS LODGED SEVERAL HUNDRED FEET BELOW TOP; OFFSHOOT OF SLIDE IN WOODS AT LEFT. ENORMOUS BROKEN BUT LODGED MASS AT LOWER RIGHT.

FIG. 5.—VIEW AT OUTER FRONT ON THE SLIDE DAM IN AUG., 1925, SHOWING SEEPAGE ESTIMATED AS 500 SECOND FEET. NOTE ABUNDANCE OF LARGE BLOCKS OF SANDSTONE AND LIMESTONE. (*Photo by Depue Falck.*)



FIG. 6.—VIEW NORTH DOWN PATH OF THE LANDSLIDE FROM HEAD OF SCARP. LAKE, DARK, AT RIGHT OF DAM. IN RIGHT BACKGROUND, ABOVE CLIFFS OF RED SANDSTONE, IS HIGH LEVEL BENCH ON WHICH IS PRE-WISCONSIN GLACIAL DRIFT.

FIG. 7.—HEAD OF LANDSLIDE SCARP. LINE OF TREES IN MIDDLE IS ON A GREAT MASS WHICH SLID DOWN ABOUT 300 FT. AND LODGED.

1925, Depue Falck, of the U. S. Geological Survey, estimated the seepage as 500 sec.-ft. On account of this seepage the lake did not overtop the dam, and when I visited it about a month later the lake level had fallen about 9 ft. below the high-water mark. Fig. 5 shows an abundance of large blocks of sandstone and limestone (5 to 15 ft. in diameter) at the outer face of the dam.

There was considerable speculation by engineers and others as to what would occur when the snow melted in the spring of 1926. It happened that there was a light snowfall in the Gros Ventre Mountains in the winter of 1925-26, so no spring flood occurred, and conditions remained practically unchanged. In September, 1926, I crossed the dam just above the point of vigorous outflow shown in Fig. 5 and examined the crest of the dam from end to end. G. E. Manger, my assistant, and I climbed the mountain slope along the east margin of the scarp (the left margin shown in Fig. 4) to its head, which my barometer indicated to be about 2100 ft. above the bed of the stream below the dam—that is, nearly four times as high as the Washington Monument. From the head, the view shown in Fig. 6 was taken looking down the path of the slide to the top of the dam and foot of the lake and to the red sandstone cliffs (Triassic) and higher hills of Jurassic, Cretaceous, and Eocene strata in the back. This shows the new auto road which has been graded across the north end of the dam and the old road below it which was buried under hundreds of feet of rock debris by the slide.

The place where the break eventually occurred is about opposite the lowest point of the lake shore. Although there was such a mass of big blocks of rock on the outer front of the dam back of this belt of rocks, the top of this part of the dam appeared to be composed very largely of fine, loose, easily erodable material, greenish-white clay, reddish clay, and sand with fewer large stones. There was a narrow sag in the crest of the dam and the high-water mark showed that the lake had extended into this sag, but I saw no evidence at that time (September, 1926) that there had been any erosion of a channel or that water had gone over the top of the dam. I was told, however, that some water had gone over the dam, but I hardly understand how it could have done so without cutting a channel through the loose fine material.

On the steep slope adjacent to the upper half of the slide scarp we found old breaks indicating that at some time great masses of rock had started to slide, but lodged. Ranchers said, however, that no other slides are known to have occurred in this part of the valley since it was settled. There have been numerous slides farther up the valley. One of these was described by Blackwelder.¹

¹ E. Blackwelder: The Gros Ventre Slide an Active Earth Flow. *Bull. Geol. Soc. Amer.* (1912) 23, 487.

Fig. 7 shows the head of the scarp. Several hundred feet below the top is a mass which started to slide, but lodged with the trees standing upon it. The declivity at the head of the scarp is 30° to 45° . Farther down it is about the same as the dip of the strata; *i. e.*, 15° to 21° . A line of sight from the top of the scarp to the lowest point of the lake dips 17° and to the farthest high point of the dam is 13° . Limestone and sandstone are exposed in the scarp, also reddish and whitish clays such as were noted in the dam. Saturation of such clayey layers by water was doubtless the cause of the big slide.

THE FLOOD OF MAY, 1927

The prediction of some engineers, and the fact that there was no break in the dam for nearly two years after the slide occurred, had given rise to a feeling of security and a rather general belief that the dam would hold. In the winter of 1926-27 heavy snows fell in the Gros Ventre Mountains and a period of rapid melting, together with rain, in May, 1927, caused a rapid rise of the lake, which, on the morning of May 18, overtopped the lowest part of the dam and caused a disastrous flood. In July, 1927, in company with my assistant, Edward F. Richards, I revisited the place to see the effects of the flood and the condition of the dam and lake. From the forest ranger, C. E. Dibble, and other residents of the village of Kelly, I obtained information as to what occurred.

We found that the outflow had cut a channel (Fig. 8) nearly 100 yd. wide and about 100 ft. deep beneath the crest of the dam near its downstream face. The channel had a rather steep gradient and the head of the outlet was considerably back of the highest part of the dam, where a shallow, but a swift stream was flowing over the rocky debris. Measurement with a hand level showed the lake surface to be about 60 ft. below the high-water mark, so that there was yet probably a maximum depth of nearly 150 ft. of water in the lake.

In 1925, Herman Stabler, of the Geological Survey, estimated that, with the basin filled to the point of overflow, there would be 164,000 acre-feet impounded and that the cutting of a channel only 25 ft. deep would release about a quarter of this great amount of water, or 48,000 acre-feet. It was then predicted that sudden release of such a flood could scarcely result otherwise than disastrously. The dam did not fail as a whole, but the lake was lowered more than 50 ft. in a very short time, and disaster resulted.

The river had been slowly rising during the morning of May 18 at the village of Kelly, about 4 miles below the dam, and efforts were being made to prevent the highway bridge from being washed out. The stream had not overflowed its 10 to 15-ft. banks there up to 10 a. m. The appearance of some ranch utensils in the stream, however, warned Mr. Dibble

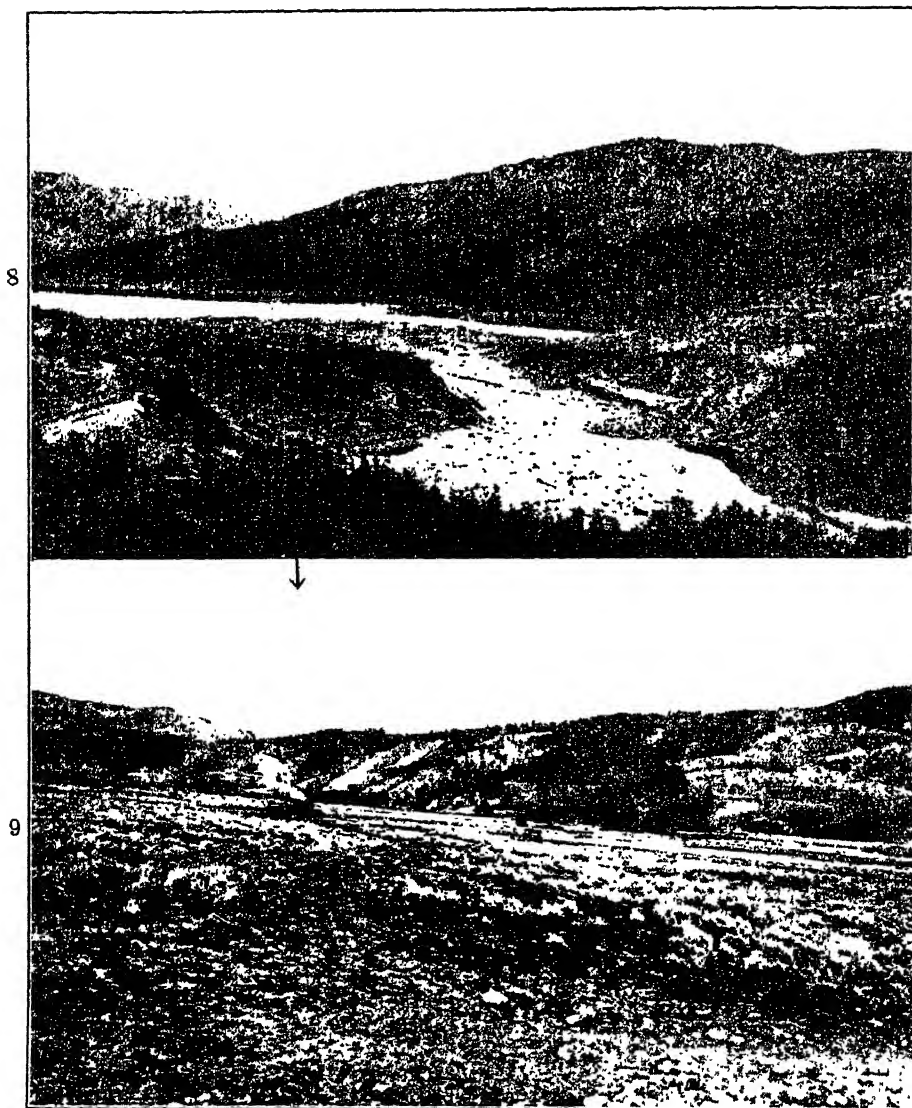


FIG. 8.—VIEW SHOWING OUTLET CUT THROUGH LANDSLIDE DAM BY FLOOD, MAY, 1927. LOW TERRACE IN OUTLET. LAKE LOWERED ABOUT 60 FT. BELOW HIGH-WATER MARK SHOWN BY LIGHT STREAK ABOVE SOUTH SHORE.

FIG. 9.—SITE OF VILLAGE OF KELLY, WYO., WHICH WAS DESTROYED BY FLOOD. BACK OF AUTOMOBILES, WHICH ARE ON THE MAIN STREET, IS MOUTH OF GORGE OF GROS VENTRE RIVER (SHOWN BY ARROW).



FIG. 10.—VIEW SHOWING ROCKS 5 TO 20 FT. IN DIAMETER, SWEEPED OUT OF CUT THROUGH LANDSLIDE DAM (MIDDLE) BY FLOOD.

FIG. 11.—VIEW OF SOUTH SIDE OF NEW CHANNEL CUT THROUGH THE LANDSLIDE DAM BY THE FLOOD. THE TERRACE WAS PROBABLY BOTTOM OF CUT MADE BY BIG RUSH OF WATER. TOTAL HEIGHT ABOUT 100 FEET.

that something was wrong at the dam. Rushing over the hills in his automobile, on the road leading up the valley, he witnessed the destruction of the buildings at Woodward's ranch about a mile below the dam and out of sight of it. He then sped back to the village and messages were sent down Gros Ventre and Snake River valleys to warn the inhabitants of the approaching flood. It is reported that about 11 a. m. there suddenly came from the mouth of the gorge above Kelly a great rush of water described as a wave 15 ft. or more in height, which caught some of the villagers as they were trying to save some of their effects. All the buildings were swept away with the rush excepting the schoolhouse and a small church and near-by cottage on a little higher level at the north edge of the village.

Fig. 9 shows the broad gravelly flat on which the village of Kelly stood at the east side of Jackson Hole. The automobiles are on the main street which led through the village. Most of the buildings were a little to the right of this view. Back of the automobiles is the mouth of the gorge from which Gros Ventre River flows from the mountains into Jackson Hole.

In 1925, Kelly had a population of 60 or 70 people, of whom a few were drowned. Besides the destruction of the buildings and their contents in this village, several ranches on Gros Ventre and on Snake River were more or less damaged. Six or seven persons were drowned, some had very narrow escapes, bridges were washed out and effects of the flood were felt as far down Snake River as Idaho Falls, Idaho. It appears that there was time after the warning was given for all persons to have escaped with their lives if they had not stopped to save any of their effects. As is usually the case, however, some apparently did not appreciate the necessity for such haste. It is reported that water began to flow over the dam in three places on the afternoon before, but it does not appear that any definite watch was maintained at the dam even after flood conditions began to develop.

That which is of most interest to geologists and engineers is what actually happened at the dam. Fig. 10 shows, in the middle ground, the cut through the dam and in the foreground the great blocks of sandstone and limestone which it had been thought by some would retard the outflow in case of an actual break. They now lie on a low terrace over which the flood rushed as it swung against the north side of the gorge below the dam. Some of these blocks are 15 to 20 ft. in diameter. They are somewhat scratched and their edges are bruised. I do not know just how far down the valley such blocks were carried by the flood. The terrace on which these rocks lie corresponds to the remnant of a terrace in the cut through the dam.

Fig. 11 is a view of the south wall of the channel. The terrace, which is 15 or 20 ft. above the stream in this view, was probably the bottom of

the channel at the time the great rush of water passed through. From this the main current swung to the north side of the valley, dumping the big rocks on the inner side of the bend, and cutting into the base of the cliffs. The narrowed but still vigorous stream flow after the big rush probably cut the inner channel below the terrace. The loose and heterogeneous character of the material composing the dam is also shown in this view. This bluff is about 100 ft. in height and it is composed of a mixture of loose sliding sand, crushed rock, and large and small angular fragments of limestone and sandstone. Evidently such a jumble of loose porous material as composes the upper part of the dump would not form a permanent dam capable of withstanding overflow without controlling gates and adequate spillways. No trees or logs were seen buried in the rock debris. Evidently the forest trees rode down on top of the slide and were not mixed into the debris very much by rolling over. They form an almost impassable tangle of criss-crossed trees, some of them still alive, on top of the dam.

Apparently no person saw just what actually took place at the dam, but the composition of the upper part of the dam, as shown in this view, may explain why the main flood came so suddenly and with such great volume. Probably seepage increased rapidly as the lake rose and this tended to undermine at the same time as overflow at the lowest point cut the initial channel. There was an enormous quantity of water in the upper 50 ft. of the impounded body. Rapid deepening of the trench across the crest must have caused rapid slumping at the sides and the loose material must have at once been swept out of the way, opening a broad outlet toward which the enormous body of water started moving with rapidly increasing velocity. Repeated slumpings of the sides may have caused the reported succession of wavelike crests of the flood. Driftwood was found 10 to 14 ft. above the ground in trees on the flat below Kelly, so that there was a considerable depth of water, though it is said to have spread out to a maximum width of about $\frac{3}{4}$ mile within a mile below the mouth of the gorge. From an instrumental survey made by L. C. Bishop, it has been estimated that there was in the upper 50 ft. of the lake body nearly 43,000 acre-feet of water, and apparently all of this went out. By 4 p. m. the flood has passed Kelly and the stream had receded within its banks.

So great is the width of the dam $\frac{1}{4}$ mile or more from front to back below the high-water level that after trenching the first 50 ft. the amount of material to be moved was enormously greater and the rate of deepening became very much slower. With the lowering of the lake level the sill of the outlet thus moved gradually back while the channel below the crest of the dam was further deepened another 40 ft. The lake was lowered only about 60 ft. before the rate of cutting became so slow as to end the flood.

Completed melting of the snow and cessation of rainfall also soon reduced the inflow so that on July 5, 1927, the lake had lowered only about 2 ft. farther below a fairly definite water mark. I have no information as to conditions since that date. Probably the lake slowly lowered through the summer of 1927 and through the past winter as the outlet channel deepened.

There was still an enormous amount of water impounded in July, 1927, for the lake extended 3 or 4 miles up the valley, nearly as far as before. It probably yet had a maximum depth of 100 ft. or more after the flood, for the tops of tall trees were emerging only near the borders of the lake.

What will happen in case there is another period of rapid melting of heavy snows in the mountains, together with much rainfall? Of course, one can not say definitely, but certainly the danger of such a flood as that of May, 1927, is greatly reduced. The outlet now provided may be deepened and broadened somewhat, but I see no reason to expect anything more than a normal high-water stage in the river below the dam. It may be that further deepening of the outlet may start renewed sliding on the south side of the valley. If this does not occur suddenly, it may simply retard the outflow by gradually encroaching on the stream, as has occurred in the case of an older slide about 15 miles farther up the valley. It would not, however, seem advisable to rebuild the village of Kelly on the same site on the low flat terrace directly below the mouth of the gorge.

GLACIAL AND INTERGLACIAL FEATURES

My brief examinations of this part of the Gros Ventre valley have yielded some unexpected evidence bearing on the history of this region during Pleistocene time, the Great Ice Age. Among the blocks of rock on the dam one well-striated block of sandstone was found. I was at first inclined to refer the scratches to scoring during the slide. Since this part of the valley has been found to have been glaciated, I think these are in reality glacial striae. About 1000 ft. up the slope on the south a beautifully striated block of rock was found in which the scratches seem certainly glacial striae.

In the right background of Fig. 6 is a dissected bench or terrace about 1000 ft. above the river channel (Fig. 2, X-Y). This high bench is partly underlaid by Eocene deposits. From the relations here and those noted farther up the valley by Blackwelder, I am inclined to think this bench represents the bottom of the valley eroded in Pliocene time. On this bench, there is glacial till, with striated pebbles, probably of early Pleistocene age. With this drift are perhaps to be correlated striated pebbles found on some of the buttes to the southwest in Jackson Hole.

Following an early Pleistocene glaciation, the region was evidently uplifted and the stream cut down another thousand feet in interglacial time. The interglacial valley is not now followed all the way to Jackson Hole by the present stream. Parts of it lie buried beneath a second deposit of glacial drift, which mantles the ridges crossed by the automobile road to the west of the landslide dam (Fig. 1).

When the flood broke through the dam it swung against the foot of the steep slope on the north and cut into the cliff below the old automobile road at the north side of the postglacial river gorge. It there exposed the interglacial gorge partly filled with quartzite, cobblestone, stream gravel cemented to conglomerate; above it is glacial till with abundant striated pebbles and boulders. Evidently the river had cut down nearly or quite as low as its present channel prior to the deposition of this second glacial drift.

I have been in this valley four times, but not until this section was exposed by the flood of 1927 did I realize that there was here an old buried valley. The line of the old valley appears to be followed by the automobile road which crosses the partly wooded ridges to reach the flat north of Kelly. Rock is exposed in the slopes both to the north and the south of this projected course, but, so far as I know, is not exposed in the slopes which lie across this course. For a short distance within the first mile below the landslide dam the river flows through a postglacial rock gorge, then it swings back northwestward into the interglacial valley which it has reexcavated at and below Woodward's ranch. After following the reexcavated valley for $\frac{3}{4}$ mile the present stream swings to the left and leaves it again to cut, through a spur of Tensleep sandstone and Madison limestone, the lower gorge from which it finally emerges just above Kelly to enter Jackson Hole. The two rock gorges between the dam and Kelly were evidently cut and the intervening section of the older valley was reexcavated after the second glaciation of the Gros Ventre valley.

Blackwelder² referred some dissected moraines farther up the Gros Ventre valley near and beyond Crystal Creek to his second, or "Bull Lake" stage of glaciation. Terminal moraines of his last, or "Pinedale" (Wisconsin) stage of glaciation are on the valley bottoms somewhat farther back. It now appears that there was glaciation at the second stage clear out to the west front of the mountains and nearly that far at the earliest, or "Buffalo," stage. Whether or not the whole valley was occupied by ice at any stage, I can not say. With my assistant, Mr. Manger, in 1926, I found evidence of a second stage of glaciation at the mouth of Flat Creek Canyon, about 5 miles south of Kelly. Nearly 5 miles farther up that canyon, on the south side of Sheep Mountain, was found the terminal moraine of the last glacial stage.

² E. Blackwelder: Post-Cretaceous History of the Mountains of Central Western Wyoming. *Jnl. of Geol.* (1915) 23, 320.

Doubtless oversteepening of the lower slopes by glaciation has had much to do with the occurrence of the numerous landslides in the Gros Ventre valley. Prime factors, however, are the presence of poorly consolidated clay shales in the upturned beds of the stratigraphic section. These clays when saturated with water may give way under the pull of gravity and either give rise to mud flows or, where covered, precipitate their overburden upon the adjacent valley bottoms.

DISCUSSION

G. S. RICE, Washington, D. C.—Those who are interested in the question of landslides in connection with mining operations should read a very excellent paper by Professor Knox.³ He classifies slides into "break-deformation slides" (such as those at the Panama Canal) and "gravity slides;" he also refers to mud flows. Professor Knox has gone into the question quite thoroughly. He has cited some cases of landslides in various countries, both naturally induced and those produced by artificial agencies, such as excavations and waste piles.

W. C. ALDEN (written discussion).—On Sept. 22, 1928, I again visited the Gros Ventre valley and reexamined the landslide dam. In general, conditions were about the same as at the time of my visit in July, 1927. I was informed by John A. Evans, road overseer of Teton National Forest, that there was a good deal of snow in the mountains in the spring of 1928 and the usual spring flood occurred, but resulted in little damage. Repeated freezing retarded melting of the snows and held back some of the outflow. I made a measurement which showed the lake to be approximately 60 ft. below the high-water mark, or about the same as in July, 1927. Whence it appears that the outlet has not been deepened appreciably during the intervening 14 months.

G. KNOX, Treforest, South Wales (written discussion).—I read with great interest Mr. Alden's excellent paper on the great landslide and flood at Gros Ventre, Wyoming. This is an excellent example in confirmation of the theory that water is the chief agent in the causation of landslides. In the diagrammatic section (Fig. 2) the Tensleep sandstone is shown resting on clay beds having a dip of $15^{\circ} 21'$ towards the valley. This sandstone is no doubt semiporous and well jointed, and receives a good supply of water from the gathering ground on Sheep Mountain. With melting snow a very large percentage of the water percolates into the strata, which probably produced a slow creep of the sandstones over the clay beds. The fissures resulting from this creep enable a still greater percentage of the rainfall or melting snow to percolate down to the impervious clays. This slow movement would upset the natural drainage of the area, resulting in the lubrication of the clay and supersaturation of the sandstone, which, combined, produced conditions so unstable that the whole mass of debris suddenly rushed into the valley forming the large dam shown on the photographs.

The Gros Ventre landslide appears to have been similar in character to the great landslide which recently took place on Mount Arbedo in Switzerland, the largest known to have occurred in that country famed for landslides. The debris displaced amounted to 30,000,000 cu. m., but thanks to the vigilance of the geologists of the Swiss Topographical Service the slow creep which usually precedes these great movements was noted and warning given to the inhabitants of the villages in the valley of Arbedo.

³ G. Knox: Landslides in South Wales Valleys. *Proc. South Wales Inst. Engrs.* (1927) 43, 161.

In this case the first movement was noted as far back as 1888, and during the following 40 years the total movement was only 5 ft. 9 in. The sliding debris finally rushed down from a height of 4000 ft. over a distance of 5000 ft., forming a dam 900 ft. high in the valley below. The holding power of this dam is exercising the minds of the Swiss engineers just as that of Gros Ventre did in your country, because should this be breached a terrible disaster is likely to result.

Further movements have recently taken place higher up the valley slope and it is expected that eventually it will reach the top of the mountain 6000 ft. high. The estimated amount of debris included in the total movement is 200,000,000 cubic meters.

In South Wales there are continual movements of the mantles on the valley slopes and although not so disastrous as the landslides referred to above are a continual source of trouble and expense to the community. Most of these valleys have been formed by the erosion of the Middle coal measures, which consists of hundreds of feet of hard sandstones known as the Pennant Series. Where the valleys are confined to this series they are steep and narrow but immediately the Lower coal measures consisting of alternating beds of sandstone, shale and fireclay—with many workable coal seams—are reached, the valleys begin to widen rapidly.

In the Middle Series the eroding action of the rivers is accompanied by slides of the "rock-fall" type, but as soon as the Lower Series is reached the slides become composite in character, consisting of the mantles formed by the rock falls of the Middle Series together with the rocks resting on the clays.

The dividing line between the two series is the No. 2 Rhondda (Brithdir, Tillery or Ynysarwed) coal seam which forms a plane of saturation on the underclay below. Downhill of this saturation plane all the soil creep or "gravity" slides take place. They work backwards (uphill) towards the saturation plane forming a series of steep crags in the hard sandstones of the Middle coal measures above the saturation plane. This leads to "break deformation" slides in the hard sandstones providing fresh material for a new mantle which in turn slides into the valley.

In these cases the debris has to slide over the outcropping edges of the strata which are nearly horizontal and in this respect differ from the Gros Ventre landslide.

As the valleys provide the only suitable place for sinking the shafts from which the coal has to be worked, they have become densely populated. Apart from the necessary buildings for domestic and industrial purposes the valleys contain all the necessary public works such as railways, canals, roads, sewage, gas and water mains, etc., so that any movement in the mantle is the cause of considerable trouble.

The cause of these South Wales landslides has been somewhat obscured owing to the fact that large heaps of colliery refuse were deposited on the hillsides during the early stages of mining development. Whenever a landslide took place in which one of these masses of refuse was included the cause was attributed to the great weight of the colliery rubbish. Failing that it was attributed to mining subsidence, with the result that many colliery companies have had to pay considerable sums for damage attributed to mining operations although the real cause was landslide movement.

The natural dam formed by a landslide such as that described in Mr. Alden's paper is always a source of great danger to the district downhill of it. The debris from the "lubricating" clays while more or less mixed up in the rock boulder mess forming the dam, will usually contain a more or less definite stratum of clay near the base, as landslides move faster at the bottom than at the top of the mass. This will act as a sliding plane on which movement may ultimately be expected and the safest plan would be to gradually breach the dam during the dry season by artificial cuttings.

Geology of the Red Lake and Woman Lake Gold Areas, Northwestern Ontario*

E. L. BRUCE,† KINGSTON, ONT.

(New York Meeting, February, 1928)

THE district of Patricia, in the province of Ontario, lies northwest of the Albany River and extends northward to Hudson Bay. Formerly this was the unorganized district of Keewatin, the southern part of which was divided in 1912 between the provinces of Manitoba and Ontario. The interprovincial boundary now runs north from Lake of the Woods for 275 miles and thence northeast 375 miles to Hudson Bay. This district is comparable in size to all the older part of Ontario south of the Canadian Pacific Railway east of Lake Huron—thus it is roughly one-third of the whole of the present provincial area.

The central part of this great triangle is one of the least accessible parts of Canada. From it streams flow southward to the English River, southeastward to the Albany, northward to Hudson Bay and the Hayes River, and westward to Lake Winnipeg. None of the streams is large and canoe travel is not easy. The only industry it has ever had is the trade in furs. Until the development of the airplane on a commercial basis, the only means of travel was by canoe or dog-train. There is not a mile of railway in the whole district.

The two areas to be dealt with lie in the southwestern part of Patricia. Red Lake is 50 miles east of the Manitoba boundary and 75 miles north of the transcontinental railway from Cochrane to Winnipeg. Woman Lake is 50 miles farther east and slightly farther north.

The late Dr. D. B. Dowling, of the Geological Survey of Canada, visited Red Lake and spent some time there during his exploration survey from Lac Seul to Berens River in 1893. He also visited the Woman Lake basin during the same journey, but his examination of it was much less detailed, the great area covered by him during the one field season making it impossible to make more than a hurried reconnaissance. Some prospecting in the Red Lake district followed Dowling's exploration, but the distance from railway was too great for any systematic work to be done.

In 1922, following the usual plan, the Ontario Department of Mines made a geological reconnaissance of the English River, following an instrumental survey of that river and its lakes by the Department of Lands. A side trip was taken to examine the rocks of Red Lake described

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by Dowling and the geological succession was found to be so promising that the following year the examination of the rocks of the Red Lake basin was continued.

One prospecting party visited the lake in 1922, and reports of the finds caused a small rush to the district in 1923, but nothing of importance was discovered and interest lapsed until the publication of the report of 1923 with a geological map of the basin. The geological succession, together with the presence of some small gold veins that had been found seemed of sufficient promise to induce the Howey brothers to make a prospecting trip in 1925. This resulted in the location of the veins south of Red Lake, which caused the enormous activity of the winter of 1925-26. During 1926, the interest spread to the areas to the northeast, including

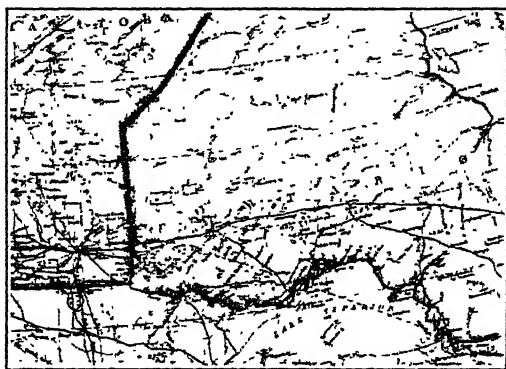


FIG. 1.—MAP OF WESTERN ONTARIO.
Rectangle No. 1—Red Lake area.
Rectangle No. 2—Woman Lake area.

Woman, Narrow and Clear Water Lakes; and, during 1927, active prospecting and development work continued in both sections of the field. Mining operations are now under way both at Red Lake and Woman Lake (see Fig. 1).

TOPOGRAPHY

The general topography of the area is that of the great Pre-Cambrian shield. The region was evidently reduced to an area of low relief prior to the deposition of the Paleozoic sediments and this Pre-Cambrian peneplain has merely been reexposed by later erosion, with comparatively little later sculpturing. The maximum difference of relief is less than 300 ft., but where the rocks are not covered by Quaternary lake deposits the erratic distribution of the glacial covering leaves a fairly large proportion of the consolidated rocks exposed. In the part of the region once occupied by the postglacial lakes, the original structure has been concealed by clays to a greater or less degree, depending on the relation of the particular locality to the shore of the ancient lake.

In areas within the lake basin, the rivers have the character of fairly mature streams meandering across a great plain, the regularity of which is broken only locally by ridges of glacial material that formed islands in the postglacial lakes. A few shallow lakes occur, impounded by these projecting glacial ridges, but the clay-mantled region is a country of rivers rather than of lakes. North of the old postglacial lakes the structure of the consolidated Pre-Cambrian rocks is evident in the pattern of the country. The schistosity of the rocks of the old complex expresses itself in linear ridges, and this characteristic is emphasized where intrusive rocks have come in along the schistosity or along the bedding of the older rocks. Thus there have been conditioned, narrow elongated bodies of water such as Narrow Lake, which is controlled by a marked E. W. foliation of the Keewatin schists and the northeast-southwest trend of Red Lake, which lies parallel to the folds of the sedimentary rocks and the intrusive sills of quartz porphyry. Where the solid rock is granite, the lakes have an irregular and equidimensional outline, as is well shown by the smaller lakes northwest of Red Lake. The scooping out of weathered material by the continental glaciers prepared basins for many lakes. Glacial dams and hollows in the glacial debris have formed others. As a result, the whole of this part of the region is covered by a network pattern of lakes of all sizes so that the rivers are merely chains of lakes connected by short river stretches in which a short rapid or fall over a hard rock layer or a ridge of glacial debris accomplishes the descent of the river in a series of steps.

GENERAL REGIONAL GEOLOGY

The consolidated rocks of the southern part of the district of Patricia are all of Pre-Cambrian age. They consist of isolated troughs of ancient volcanic rocks and sediments, remnants of closely compressed synclines remaining as roof pendants in the great granite batholith, which probably underlies much if not all of the Pre-Cambrian shield and which is the surface consolidated rock over much of it. In some parts of the region paragneisses are abundant. These are garnetiferous and staurolitic, biotite, quartz rocks that were originally arkoses. It is assumed but has not as yet been definitely proved that they are the oldest exposed rocks. In the Red Lake and Woman Lake districts the gneisses do not occur, and the oldest rocks are lavas, mostly basic and now severely altered. In the Red Lake area the lavas are overlaid by sediments, correlated tentatively with the Temiskaming. These older rocks are intruded by a series of acidic igneous rocks which have been correlated with the Algoman and in both areas there are dioritic rocks the relation of which to the postvolcanic sediments is not known.

With the exception of small lamprophyre dikes, no consolidated rocks are known later than the Pre-Cambrian acid intrusions, but it is possible that, after erosion had reduced the Pre-Cambrian complex to low relief, Paleozoic rocks may have been deposited, connecting the Ordovician, Silurian and Devonian rocks which occur in the James Bay basin with similar formations in Manitoba. If so, they have been completely removed and the ancient Pre-Cambrian surface exposed.

The continental glaciers removed from the consolidated rocks whatever weathered material may have accumulated, and during the ice retreat, discontinuous and unsorted deposits of glacial debris were left behind. At one stage in the retreat, the ice front seems to have stagnated along a line running northwest across the east end of Red Lake, and thence southeast a few miles south of Woman Lake. The stagnant ice formed the northerly shore of a postglacial lake in which clays were laid down during a fairly long period. Since the retreat of the ice there has been merely the slight erosion of rivers on this old lake plain.

GENERAL GEOLOGY OF RED LAKE BASIN

The pre-granite rocks of the Red Lake basin form a trough approximately 30 miles in length with a maximum width of 8 to 10 miles. The volcanic rocks, which have been called Keewatin, consist chiefly of basic types now altered to massive greenstone or to chlorite schists. In some localities, amphibolitic rocks are abundant, possibly as contact types, but highly serpentinous and talcose rocks are prominent varieties. As all of the rocks are so highly altered, it is unsafe to attempt to give any specific names to the original lava types. Both basalts and andesites commonly alter to chloritic rocks, which are best classed simply as greenstones. Rhyolites are less abundant than are the more basic varieties but they are useful in unraveling the structure of parts of the basin.

The volcanic rocks were intruded by diorites which truncate the flows and which tongue out upward into the Keewatin rocks. This is taken to be evidence that they were intruded after the folding of the lavas. The relation to the sediments is not known but at present they are assumed to be older.

The sedimentary beds correlated with the Temiskaming consist of graywackes, arkoses, and slates, with lenses of conglomeratic material. They are evidently the product of terrestrial deposition. They lie unconformably above the volcanic rocks.

Economically, the acidic intrusives that have been grouped together as Algonian are by far the most important of the Red Lake rocks. This intrusive period, if it be one period, began with immense invasions of quartz porphyry into the old metamorphics. Where the porphyry

intruded sediments, it entered chiefly as sills, but in the massive pre-sedimentary rocks it took the form of dikes or bosses. In character the porphyry is extremely variable: the phenocrysts may be abundant or may be almost lacking; further, they may consist entirely of resorbed quartz crystals; they may be quartz and orthoclase or plagioclase or they may be entirely of plagioclase. These variations perhaps represent variations in composition produced by absorption of masses of intruded rock.

The porphyry itself has suffered considerable alteration and sericite makes up a large part of the groundmass. Carbonates also are abundant. The intruded rocks have suffered very severe alteration, especially near quartz-porphyry tongues, which plunge beneath the older rock. In such places the greenstones have been extensively altered to ankeritic material. Calculations show that this probably takes place with decrease of volume and fractures develop which are later filled with stock works of quartz. The effects upon the sediments seem to be much less marked.

The quartz porphyry was possibly the forerunner of the immense intrusions of granite which surround the older rocks, and in two places cut up through them as bosses. The larger of these bosses is the elliptical mass which occupies the central part of Red Lake. In area it approaches the lower limit set by Daly for the application of the term "batholith."

The granite, whether of the main mass or of these smaller intrusions, is the typical rock consisting of quartz, orthoclase and plagioclase with either biotite or hornblende as the chief dark-colored constituent. In the main mass the rock is fairly fresh but there are large areas of granitic rocks, amphibolites and quartz-biotite gneisses, which lie between the main granite batholith and the greenstone or sediments, and appear to be border zones produced by the early invasion of the granite magma. There was incomplete assimilation of included rock, and the production of a heterogeneous hybrid magma, later crystallized as a dark gray to light gray porphyritic granite. In other places a rock was produced which retained the foliated structure of the intruded rocks, but with a composition profoundly altered by the *lit par lit* injection of granitic material. Later advances of the main magma destroyed parts of these zones and sharp contacts are usual between granite and the hybrid rocks.

STRUCTURE OF PRE-CAMBRIAN ROCKS AT RED LAKE

The structure of the Pre-Cambrian rocks is as yet imperfectly known, and the following summary must be taken as only a general statement which will necessarily be altered in detail as more information becomes available. The Keewatin rocks, after their consolidation, were thrown into close east-west folds which still pitch easterly. These folded rocks were intruded by bosses, dikes and sills of diorite. Over these rocks and

separated from them by erosion sufficiently deep to expose the diorites, beds of heterogeneous sediments were deposited. After their consolidation, they were compressed once more into eastwest folds which now plunge westerly. The impression of a westerly plunge on these beds must have reduced the steepness of the easterly plunge on folds in the Keewatin rocks. An anticline of Temiskaming sediments occupied the position of the main axis of Red Lake and this was flanked on the north by a syncline, the axis of which lies across the central part of Slate Bay, and on the south by a syncline occupying the southern part of Mackenzie Island. The northern limb of the northern syncline has been removed by the granite intrusion and the southern limb of the southern syncline by the quartz porphyry and granite intrusions of that area. One remnant of sediments south of the lake shows that other synclines once existed in that direction.

GENERAL GEOLOGY OF WOMAN LAKE AREA

The geology of the Woman Lake area differs in some important respects from that of the Red Lake basin. The oldest rocks are similar to those at Red Lake in being basic volcanic flows with a minor amount of rhyolitic rocks with which are interbedded some tuffs. At Woman Lake there are large areas underlaid by volcanic agglomerates.

The lavas are intruded by great masses of basic igneous rock of somewhat variable type to which the general term "diorite" may be applied. In some occurrences, the rocks are holocrystalline and dioritic. In others the rock has comparatively little feldspar and becomes an amphibolite. Near the margins the texture is extremely fine and the diorite is distinguished with difficulty from the lavas of the previous period. In these marginal facies a porphyritic habit is commonly developed and the rock becomes a black quartz porphyry. Since there are among the lavas certain types in which quartz phenocrysts also occur it is possible to distinguish the diorites from the lavas only by determining the relationship to other rocks. Where the rock becomes schistose it is practically identical with the Keewatin chlorite schists.

The diorites commonly occur in elongated elliptical masses. They are resistant and form ridges. Woman Lake owes its north-south elongation to the north-south ridge of diorite that forms the divide between Woman and Clear Water Lakes.

Sediments that can be correlated with the Temiskaming rocks of Red Lake are present in the immediate vicinity of Woman Lake, Narrow Lake and Clear Water Lake, only as a small remnant of jasper conglomerate on the west shore of Woman Lake and as a narrow trough of conglomerate at Sundown Lake northeast of Clear Water Lake.

The intrusions that correspond to those grouped as Algoman in the Red Lake district are numerous and the variety in lithology is even

greater. There is no area of quartz porphyry as large as the masses at Red Lake, but there are a great many dikes of rock of that type. In addition there are many small bosses and dikes of quartz-feldspar porphyry and there is evidence of some differences of age even among these porphyries. A large band of a granitic porphyry occurs on the islands of the western part of Clear Water Lake and strikes inland at the northern end of the lake. This rock intrudes the diorite and is believed by Greig¹ to be earlier than some of the other acidic porphyries. There seems little doubt, however, that all of these porphyries were intruded in the early part of the Algonian period, and are merely forerunners of the batholithic intrusion of granite which was the last important event of Pre-Cambrian time in this district. The porphyries represent merely the highly differentiated and abnormal apophyses of the main intrusion.

The granite is not at all different from the granite of Red Lake and is no doubt a part of the same great batholith. It intrudes all of the rocks just described and cuts even the porphyries, which are assigned to the same magma as is the granite but which had cooled prior to the advance of the main intrusive mass.

A few basic dikes occur. Some of these intrude the granitic rocks. They seem to have no bearing on the ore deposits.

THE GOLD DEPOSITS OF RED LAKE

The claims of the Howey gold mines and the adjacent claims held by the McIntyre Mining Co. are the only ones at Red Lake on which development has been carried to the stage of proving ore. The holdings of both companies lie along a quartz-porphyry dike lying a short distance south of the outlet bay of Red Lake. The dike is nearly vertical and strikes slightly north of east. The width varies from 100 ft. near the west end of the exposed part to 30 ft. farther east. It has now been traced for a total distance of 1800 feet.

The dike itself is a fine-grained rock of a light yellowish-green color and a waxy luster. Quartz is visible to the naked eye as rounded glassy blebs. The country rock is dark-colored lava. About 1400 ft. northwest of the point where the dike crosses from the Howey claims to the McIntyre claims to the west is the edge of the boss of granite, and the strike of the dike is such that it would be cut off by the granite, approximately 1600 ft. west of this point (Fig. 2). It has been suggested that the dike is not a simple one, but is actually a series of lenses arranged in echelon.² If that proves to be the case, offsets to the southward in proceeding westward would prolong the possible length in that direction.

¹ Ontario Dept. of Mines *Rept.* (1927) 36, Pt. 3, 100.

² D. G. H. Wright: The Red Lake Gold Area. *Engng. & Min. Jnl.* (1926) 122, 15.

Not all of the dike is ore, but parts of it have been severely fractured and some of the fractures have been occupied by auriferous quartz veins. Few of these veins are wider than 1 ft. and they are all lenticular in character. The gold is confined almost entirely to the quartz, but in part of the dike the veins are so closely spaced that it is believed the whole dike can be treated over minable widths. The fractures have evidently been produced by a shearing movement during which the brittle quartz porphyry was fractured and spaces opened up for the introduction of the auriferous solutions. Along the veins the porphyry shows some hydrothermal alteration, the original feldspars being changed to sericite and carbonate. In places where the quartz veins are closely spaced, the quartz porphyry may be entirely sericitized or carbonated.

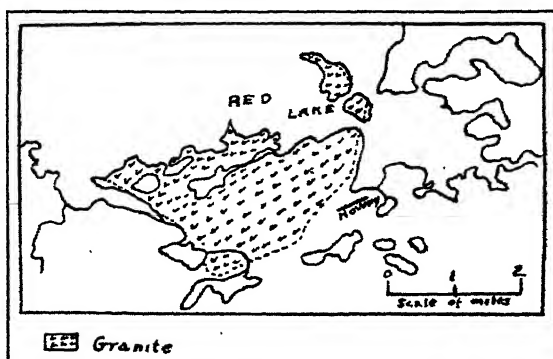


FIG. 2.—RELATION OF HOWEY VEIN TO GRANITE.

The quartz veins include in minor quantities the non-metallic minerals ankerite, scheelite, tourmaline and feldspar, and the metallic species pyrite, galena, chalcopyrite, sphalerite, tetradymite, gold and probably gold tellurides. In the porphyry along the quartz veins, pyrite is practically the only metallic mineral.

An examination of the minerals of the Howey orebody show that quartz, tourmaline, feldspars, and probably some pyrite were deposited in the fissures of the quartz porphyry at a fairly high temperature. Whether gold was present in this stage of mineralization is not known but it is clear that most of the gold came with a later period of mineralization during which fractured quartz and pyrite were impregnated by galena and sphalerite. From these facts, it is evident that the important mineralization did not come from the quartz-porphyry intrusion, since that rock shows some hydrothermal alteration by the early quartz deposition. It is clear also that the vein solutions must have come from a source high in silica and probably the alkalis. The only formation meeting the requirements of an age subsequent to the porphyry, and of an acidic composition, is the granite. If it was the parent body from

which the ore deposits came, it must have been in the later stages, since a few small gold-bearing veins have been found in granite. The succession of events so far as the orebodies are concerned may be summarized as follows:

1. Intrusion of the dikes of quartz porphyry.
2. Development of fractures in this brittle rock by an east-west shear possibly originated by disturbances inaugurated by the granite intrusion.
3. Filling of some of the fractures by quartz veins at high temperatures.
4. Refracturing of these veins and deposition of galena, sphalerite, tellurides and gold from solutions given off in the last stages of the cooling of the granite.

Other properties of some merit have been located in the Red Lake basin but none of these has reached a stage of development where gold production seems assured.

DEPOSITS IN THE VICINITY OF WOMAN LAKE

Gold-bearing veins in the vicinity of Woman Lake (including Clear Water Lake to the east and Narrow Lake to the west) are of several

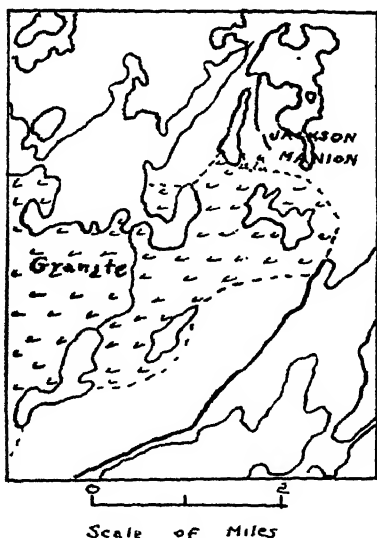


FIG. 3.—RELATION OF JACKSON-MANION VEIN TO GRANITE EXPOSURES.

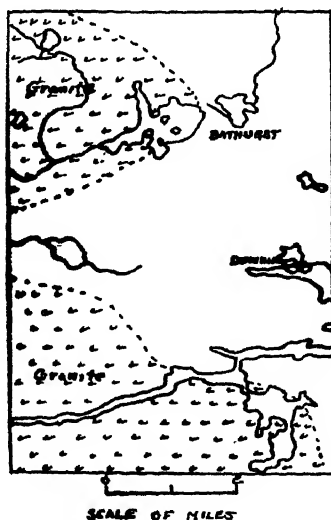


FIG. 4.—RELATION OF BATHURST AND DUNKIN VEINS TO GRANITE EXPOSURES.

different types and all differ from the Howey-McIntyre deposit at Red Lake. They are similar, however, in that the more important of them are in the neighborhood of intrusions of granite (Figs. 3 and 4). At Woman and Narrow Lakes, the granite masses have the shape of salients

projecting from the main batholiths into the older rocks, but had erosion been less profound these might have been bosses similar to those at Red Lake. Gold-bearing veins have been found in the rocks, which are older than the Algoman porphyry and granite period; the character of the veins differs with the type of rock in which they lie.

The Jackson-Manion Mine

The Jackson-Manion claims extend southward from Rowe Lake and lie $\frac{1}{4}$ mile east of Woman Lake. This was one of the early discoveries of the area and is the first property on which actual mining operations have been begun.

The predominant rock is massive greenstone formed by the alteration of lavas of medium basicity. A few dikes of diorite intrude the green-

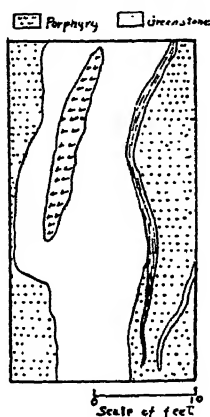


FIG. 5.—LATER BANDED VEIN ALONG WALL OF WHITE-QUARTZ VEIN.

stone, probably tongues from a great lenticular mass lying north of Rowe Lake. There are also many dikes of quartz porphyry and quartz-feldspar porphyry that have a general northerly strike. The southern part of the claims is occupied by a wedge-shaped mass of granite that tapers out to a narrow tongue northward.

The vein lies approximately 650 ft. east of the northern tip of the tongue and can be traced northward as far as Rowe Lake. Southward the vein is not continuous but there are at least two quartz lenses, one exposed and one indicated by drilling, arranged in echelon with the vein and offsetting to the east along an arc which curves to the south eastward. Granite outcrops 300 ft. directly along the strike of the southern lens, but it is possible that an offset to the eastward may make room for a third lens beneath the swamp, east of the granite outcrop. The most southerly known lens has been well exposed in development work.

At surface it has a length of 335 ft., and reaches a maximum width of 13 ft. At the north end it curves sharply to the eastward and

apparently pinches out, but deep cover in that direction has prevented thorough examination. The wall rock is fairly massive greenstone cut by narrow porphyry dikes. The vein cuts across the dikes at a small angle and includes fragments of them. The structure is not simple. The main part of the vein consists of rather milky white quartz, but on the east or hanging-wall side (the dip is steep to the east) there is a vein with dark-colored streaks parallel to its walls, which appears to be of later age than the white quartz (Fig. 5). It is somewhat sinuous but can be traced from end to end of the lens. Along most of its course it lies between white quartz and country rock but in places it leaves the main lens and cuts out into the hanging wall.

Spectacular occurrences of gold are found in both types of quartz, and in addition, petzite is present in some quantities. Some of the dark streaks in the secondary vein seem to be thin smears of the telluride along fractures in the quartz. Some pyrite is present both in the quartz and in the wall rock but the total amount of metallic minerals is small. Thus most of the gold content is in the form of native gold or as the telluride.

The relation of the vein to the granite tongue and its evident subsequence to the porphyry dikes are evidence that it is probably directly related to the intrusion of the granite. The process of vein filling was a complex one, and a fracturing subsequent to the first quartz filling conditioned the main introduction of gold values. Gold occurs in plates $\frac{1}{2}$ in. in diameter in apparently unfractured white quartz, but it is likely that the narrow secondary vein will prove to be richer than the white quartz on account of its content of tellurides.

The Bathhurst Veins

The Bathurst claims lie northwest of Narrow Lake, approximately 12 miles northwest of the Jackson-Manion. The geology is somewhat similar. Altered lavas are the oldest rocks and these are intruded by bosses of diorite of considerable size. Porphyry dikes of at least two ages cut both these rocks, and all are cut by massive coarse-grained pink granite which enters as a blunt salient from a great area of granite to the northwest.

The veins are of two distinct types. The most promising one occurs in a felsite or porphyry dike, 10 to 20 ft. wide, which cuts diorite and greenstone. The dike has been fractured longitudinally and the fissure filled by gold-bearing quartz. The width of quartz varies from 3 ft. at the widest part to a few inches where narrowest. Intrusions of a pinkish porphyritic rock cut laterally and as pipes into the felsite dike and apparently into the vein, but this latter relationship is not very clear. Gold occurs in visible quantities in the quartz. Arsenopyrite in well formed crystals, pyrite and chalcopyrite are present in small amounts. A

noteworthy fact is that the gold values are highest near the late pink porphyry. Since the vein is apparently older than the porphyry, this enrichment must be explained as an impregnation by solutions from the porphyry, following a secondary fracturing.

Some veins on the Bathurst property lie wholly within the greenstone. The original discovery consists of an apparent width of 15 ft. of white milky quartz in which are spectacular occurrences of gold. One plate is $1\frac{5}{16}$ in. by $\frac{3}{4}$ in. and still retains the groovings made on it by the glaciers. On development, it has been found that the true width of the vein is considerably less than the apparent width and that it is in reality an anticlinal fold which pitches at an angle of about 30° to the westward. Veins of this type probably conform to the bedding of the volcanic flows and perhaps are of the nature of saddle veins at the crests and troughs of the folds.

The Dunkin Claims

The Dunkin claims lie 3 miles south of the Bathurst on a narrow tongue of land projecting eastward into Narrow Lake. The country rock is diorite which has suffered considerable schisting and, at a period later than the schisting, was cut by a porphyry dike 10 to 12 ft. wide. The dike cuts across the foliation at an angle of 30° . The vein is parallel to the foliation of the rock and so cuts the dike. The quartz is white and milky along the greater part of the vein, but at the intersection of the vein and the dike the quartz is deep blue in color. Gold is visible at various parts of the vein, but is exceptionally abundant in the blue quartz.

This deposit has not been developed to the same stage as the Jackson-Manion and Bathurst, and hence fewer details as to its structure are available. The striking difference is the fact that it lies near no visible granite to which it can be assigned.

The Tivy-Sloan Vein

Other deposits of varying prospective merit might be mentioned but few have as yet had much work done on them. As another type of deposit it is worth while describing the remarkable vein on the Tivy-Sloan claims which lie east of East Clear Water Lake, 7 miles east of the Jackson-Manion.

This vein is remarkable for its length and uniformity. It has been stripped absolutely bare, with the exception of a couple of hundred feet for $\frac{5}{8}$ mile. It has been found again to the south across a swamp and a bay of the lake and traced for another 650 ft. and then, beyond an area of heavy drift cover it has been uncovered for 900 ft. At the north end the strike is nearly north-south. To the southward it curves in a gentle

arc to a strike somewhat west of south. This arc, projected across the lake, joins an arc with a more southerly strike, and this arc, projected across the drift-covered area, connects with the southern exposure, with a southwest strike.

The width of the vein is remarkably uniform. The lenticular character so common in quartz veins in Pre-Cambrian rocks is absolutely lacking. At the north end as exposed, the width is 3 in.; the middle part of the vein is $2\frac{1}{2}$ ft. wide and the southern arc reaches a width of 6 to 7 ft. of quartz. The appearance in all parts is similar, the vein showing a decided banding and a wavy or colloform structure. The dip of the northern arc is 75° W.; of the southern arc, 75° N. W. The quartz contains a little pyrite and some gold is visible but the occurrences are less spectacular than those of the Bathurst, Jackson-Manion and Dunkin deposits. Black tourmaline is conspicuous on fracture planes in the southern part of the vein.

The country rock is ellipsoidal lavas, with fragmental flowtops well developed. The vein occurs in or near one of the fragmental tops, and the gently curving course of the vein probably represents the bedding of the ancient flows. The only other rocks that occur are narrow lamprophyric dikes which cut the veins and have no connection with the mineralization.

For this deposit, therefore, there is lacking not only any granite tongue or boss in the vicinity but also any visible intrusion preceding the formation of the vein. It appears, then, that veins may occur under diverse conditions and any generalization at the present stage of development is hardly justified.

It can be said, however, that the larger and richer of the deposits so far discovered are in the neighborhood of salients of granite which is believed to have furnished the auriferous solutions, and they are in rocks that have been previously intruded by porphyry dikes which no doubt shattered the older rocks and provided larger and more continuous channels for ore deposition than would otherwise have been the case.

CONDITIONS OF DEVELOPMENT

The earliest mining development in the Red Lake district began in 1895, when the only means of access was by canoe from the Canadian Pacific Railway 100 miles to the south. Nothing came of this venture, and even after the construction of the Canadian National Railway had made the Red Lake district more accessible, the distance seemed to be too great for successful development of mineral deposits. Therefore, after geological mapping had been completed in 1923, it seemed unnecessary to undertake the exploration of the Woman Lake area still farther from the railway.

Conditions of travel, however, have been changing rapidly in northern Ontario. The first step in this modernization of prospecting was the

introduction of the outboard motor, which reduced the labor in canoe travel and led prospectors to undertake longer journeys. The next step was the introduction of air travel as a rapid and easy means of reaching hitherto inaccessible areas. The result has been the unprecedented rapidity with which both the Red Lake and the Woman Lake areas have been opened up.

During the rush for location of claims that follows every promising discovery, airplanes have made possible extremely rapid staking. Equipped with floats, the machines are able to land on any lake of fair size; hence, in a country that is at least one-third water, it is possible to come within a mile or so of any point. As development has proceeded, the air service in the north country has been of immense utility. Even during winter months the machines, equipped with skis for landing on snow, have continued regular flights both to Red Lake and to Woman Lake. At first the chief service was in the carrying of mail and of passengers to whom time was a consideration. During the past season especially, at Woman Lake, air transportation from Lac Seul, the end of easy water navigation, has competed successfully with water transportation of freight. Supplies could be laid down at any given place in the district nearly as cheaply and certainly in better condition than they could be brought in by canoe. Many prospects some distance off the main water routes would never have been examined had it not been possible for supplies to be laid down right at their own lake front.

The development of Red Lake and Woman Lake areas has shown that with modern methods no part of northern Ontario is too far away from regular transportation systems to be considered as a prospective mining area. Development can be carried forward by air service at reasonable costs to a point at which it can be decided whether a deposit is of sufficient value to warrant the building of a railway.

DISCUSSION

G. A. PACKARD, Boston, Mass.—Are the occurrences of gold in these rocks supposed to be related more particularly to structural causes or are they influenced more by the composition? For example, I have a report on property on the west side of Pipestone Bay. The report states that what is called quartz-porphyry differs in character, part of it being rather light colored, quite acid, and the other portion being slightly darker; and also that the gold has been found in the auresite rather than in the apparently more acid rock. The writer believes that this is not due to the variation of composition but to structural details.

E. L. BRUCE.—I presume you mean when gold occurs in the vein, because it seems rather general in this district that there is comparatively little impregnation of the wall rocks by the solutions.

When we speak of the Howey as an orebody we only mean that the whole dike must be mined in order to get the quartz. It is structure rather than a difference in the composition, because if it were a difference in the porphyry we would expect some

indication of it. Quartz porphyry is only a general term; it is impossible to make minute divisions to cover all variations.

D. T. O'CONNELL, New York, N. Y.—On Mackenzie Island in Red Lake may be found a vein that apparently looks like pure quartz but actually contains tellurides, and I would like to know if you happen to have come across that and what the relation is between that particular kind of vein and the normal gold pyrite-bearing vein.

E. L. BRUCE.—I am sorry that I have not seen that vein on Mackenzie Island. How much telluride is there?

D. T. O'CONNELL.—I can only say that it is a telluride vein, because I happened to throw some pieces in the fire, and in a few minutes a few specks of gold appeared. A solid section across the vein assayed \$10 gold to the ton.

E. L. BRUCE.—I think that vein is probably exactly similar to the others except for a variation in the mineralization.

D. T. O'CONNELL.—There is no pyrite visible.

E. L. BRUCE.—In the Howey the amount of pyrite is small. The difference perhaps depends on difference of temperature. In other words, if the Howey vein were traced toward the granite contact, it might be found to contain tellurides instead of pyrite.

D. T. O'CONNELL.—The vein I found was at right angles to the strike, and it seemed as if it were a different sort of vein, entirely, from the Howey. It have seen the Howey. They had a different bed-rock or wall-rock composition in the Howey, located between greenstone and quartz porphyry, whereas this is what I would call a telluride quartz vein and was located between slates, or what appeared to be slates.

E. L. BRUCE.—In that district you are fairly close to the granite boss, and I rather suspect that it is the proximity of the granite that determines the character of the mineralization.

G. W. BAIN, Amherst, Mass.—I happen to have sampled and examined the veins on Mackenzie Island at the invitation of Timothy Crowley, the owner, and none assayed as high as \$10, so that Mr. O'Connell's communication comes more or less as a surprise. One did run \$9.60 but rechecks taken on new samples failed to assay over \$0.80. The veins in one section of the island contained a mineral that resembled telluride but on further study turned out to be arsenopyrite.

The telluride, tetradymite, has been reported from the Howey property. I was never able to find it but did encounter another with similar physical properties which could be confused with tetradymite unless the chemical tests were carefully made. I picked half a gram of this mineral from one of the McIntyre veins; the crystals were not very good but they were as good as anybody could get. Analysis showed that it was a lead sulphotelluride with a trace of gold and silver and fitted the formula $(\text{Pb}, \text{Ag}, \text{Au})_2(\text{S}, \text{Te}, \text{Se Sb})_3$ for nagyagite. Careless tests would confuse the lead with bismuth. Krennerite also occurs but is very rare.

D. T. O'CONNELL.—May I say a word about the remarks of the previous speaker? He mentioned a vein on Mackenzie Island. That island is $4\frac{1}{2}$ miles long and in parts it is $1\frac{1}{2}$ miles wide. There are many veins on the island. In fact, we took samples of nine veins, ourselves, of different kinds. The vein I have in mind could hardly be called a vein. It is a veinlet. It was only 3 in. wide, and we blasted the entire veinlet out and took it with us. At least, we blasted the entire top out of it and went down the dip vertically into the ground a few feet.

In regard to being mistaken about the gold content: I roasted the ore right on the spot and found the specks of gold coming out, to my great surprise, because it was the

first gold I saw on my whole trip to Red Lake. We brought the samples of this ore back to New York and they were assayed by Ledoux & Co.

A. M. BATEMAN, New Haven, Conn.—I should like to ask Mr. Bruce if he saw, in any of the quartz veins of this section, any indication that they might be vein dikes?

E. L. BRUCE.—The only thing that could be called a vein dike—or my idea of a vein dike—is the quartz of the first generation, the quartz that carries some pyrite and comparatively no gold. It has been fractured and the fragments included in the secondary vein, which carries most of the gold.

G. W. BAIN.—I do not know whether the vein structures to which Mr. Boydell has drawn our attention are general throughout northern Ontario but they are characteristic of the Howey-McIntyre deposit at Red Lake. Over 50 specimens containing gold, representing the majority of localities where values were found, showed that the gold and tellurides filled late stage fractures in every case; and conversely, other thin sections showed that veins lacking strong secondary fractures were consistently barren.

The age relation of the granite and the so-called quartz porphyry does not seem to have received the attention it deserves. One of the many rock types designated as quartz porphyry is exposed in the west central part of the McIntyre group of claims and grades into the granite batholith outcropping north of the Howey, from which it is evidently a marginal offshoot. Dr. Bruce, Dr. Hawley, Mr. Wookey and myself went over the east end of this exposure in detail because, to the McIntyre staff, it seemed to be of great geological significance.

The exposure ends in a cliff about 20 ft. high and 40 ft. wide. A black dike-like mass about 20 ft. wide protrudes upwards from the central base of the cliff and at 12 ft. upends in a series of jagged fingers projecting into the quartz porphyry cover. Narrow shear planes marked by greenish sericite extend upward from the top of some of the fingers and even from the main part of the dark mass, which is slightly sericitized irregularly throughout and not along localized planes. Localization of sericite in the overlying rock in contrast to general, irregular distribution in the underlying, and the fact that the dike-like mass widens greatly downward, seem to indicate that the latter is intrusive into the porphyry cover. However, Dr. Bruce saw these features and dissented from our opinion.

The dike-like mass is petrographically similar to the so-called porphyry at the Howey, but I cannot regard the rock as a true quartz porphyry; most, if not all, the quartz in the rock associated with the gold deposits on the McIntyre and adjoining section of the Howey is secondary and seems to have been introduced at a stage when the fracture systems had not become as highly localized as the chief ones now appear. The rock appears, on microscopic evidence, to be an andesite.

E. L. BRUCE.—Dr. Hawley and I could not see it in that way. I am willing to admit that when it comes to the intrusion of one kind of igneous rock by another it is sometimes extremely difficult to tell which is the intruding rock. And the older the rocks, the more difficult it is. So there is a difference of opinion between Mr. Bain on the one hand and Mr. Hawley and myself on the other, and if you want to decide that, you will have to go to see the outcrop yourself.

H. C. BOYDELL, Cambridge, Mass.—Is that feature developed to any great extent in the area?

G. W. BAIN.—That is the only one I know above ground.

E. L. BRUCE.—Mr. Bain also said that the Howey dike is an andesite. I cannot agree. I am doubtful if the term *andesite* is applicable in such a relation. The quartz there is phenocrystic, in my estimation. There again Mr. Bain and I differ.

Sedimentary Metalliferous Deposits of the Red Beds

By JOHN WELLINGTON FINCH,* DENVER, COLO.

(New York Meeting, February, 1928)

IN AUGUST, 1927, the writer examined certain copper deposits in New Mexico¹ which occur in beds of sandstones and shale, and in connection therewith reviewed the literature upon deposits of this type. Such deposits are known and have been reported upon at various points in Texas, Oklahoma, Arizona, Colorado, Wyoming and Utah, but in New Mexico they have been given more development than elsewhere and apparently occur there more conspicuously than in the other states.

Some 25 years ago the writer examined deposits of similar origin in northern Arizona and more recently other deposits in western and southwestern Colorado, and these examinations have led him to realize how widely scattered are the deposits and how generally noncommercial they are. On superficial development, to one without a comprehension of their geologic nature, they are alluring on account of occasional small amounts of high-grade ore and much misplaced capital has been expended upon them. This paper, therefore, aims to present briefly certain theories of genesis, some details of which may possibly be new, and to lead up to practical conclusions which may be useful to other geologists and engineers.

A bibliography of the more important literature upon such deposits is given below.² Coffin's *Bulletin* upon the Radium, Uranium and Vanadium Deposits of Colorado, published in 1921, is a particularly clear discussion. The most comprehensive recent statement is Butler's.

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¹ Localities in New Mexico are given in U. S. Geol. Survey *Prof. Paper* 68, 76.

² W. Lindgren, L. C. Graton and C. H. Gordon: Ore Deposits of New Mexico. U. S. Geol. Surv. *Prof. Paper* 68 (1910).

S. F. Emmons: Investigation of Metalliferous Ores. U. S. Geol. Surv. *Bull.* 260 (1904).

W. Lindgren: Replacement Orebodies and the Criteria for Their Recognition. *Econ. Geol.* (1911) 6, 556. Also Textbook upon Mineral Deposits (1919) 399-412.

A. F. Rogers: Origin of Copper Ores of the "Red Beds" Type. *Econ. Geol.* (1916) 11, 366.

W. H. Emmons: The Cashin Mine, Montrose Co., Calif. U. S. Geol. Surv. *Bull.* 285, 125.

W. A. Tarr: Copper in the "Red Beds" of Oklahoma. *Econ. Geol.* (1910) 5, 221.

B. S. Butler, G. F. Loughlin and V. C. Heikes et al: Ore Deposits of Utah. U. S. Geol. Surv. *Prof. Paper* 111 (1920).

R. C. Coffin: Radium, Uranium and Vanadium Deposits of Southwestern Colorado. Colorado Geol. Surv. *Bull.* 16 (1921).

He describes a large number of occurrences in Utah and connects them up genetically with deposits in other states.

Deposits of this type are particularly characteristic of the "Red Beds." Members of the so-called "Red Beds" group have been assigned to periods ranging from upper Carboniferous to early Cretaceous. At many points in the Rocky Mountain region of the United States, Permian, Triassic and early Jurassic have the appearance of a stratigraphic unit. In certain areas they are conformable throughout and exhibit continuity of conditions of climate and deposition. Jurassic and Triassic beds have much the same distribution. The exact age limits of the beds are somewhat immaterial in the present discussion. The metalliferous strata are found locally at all horizons in nonmarine Red Beds. In all cases these deposits lie in beds containing plant fossils. These constitute the Jurassic and Triassic formations in the regions to be discussed below, and possibly certain underlying beds whose age has not been fully agreed upon.

ORIGINAL CHARACTER OF DEPOSITS

Briefly, the essential conceptions are as follows: Following the Carboniferous coal measures, there was a considerable climatic change, particularly a decreased humidity; the shallow seas receded during the Permian from a broad uplift extending from central New Mexico into western Wyoming in a direction somewhat more northwesterly than the present Rocky Mountains; during this time the Carboniferous and earlier Paleozoic beds were being removed from this land area; and by the beginning of the Triassic as generally defined, Pre-Cambrian outcrops were supplying much of the materials of the Red Beds. Erosion had proceeded to such a point that there was a low continental divide along which Pre-Cambrian rocks appear to have been exposed more or less continuously. Erosion products from this ridge were spread east and west over broad flood plains through an average distance of, roughly, 200 miles on its west flank in northern Utah and Colorado, and possibly 300 miles in northern Arizona and northern New Mexico; and for a distance more difficult to determine on the east flank of the ridge. On both sides the Red Beds contain gypsum locally and lime generally though the beds are predominantly of clastic materials. This continental area was bordered by advancing and receding seas. On the west enormous thicknesses of Triassic and Jurassic marine beds were deposited in Nevada and California, but thin beds in the intermittent epicontinental sea to the east, which advanced toward the end of Jurassic, depositing marine beds in New Mexico overlapping the Red Beds into the region of the present mountain range.

Since the metals were derived from weathered outcrops, they were transported either as fragments of minerals from the zone of oxidation or as soluble salts, generally sulfates. The transported particles of iron and

manganese were oxides and carbonates, and the mineral fragments from copper veins were mostly carbonates. Under such conditions of land deposition in a semi-arid climate,³ streams did their most important transportation at times of violent local precipitation. They also deployed about over their flood plains, and changed their courses rapidly. As a result the beds of sand and gravel thicken and thin out in a most erratic manner. It also appears that the particles of metallic minerals were carried from the hills area only intermittently, probably when local heavy precipitation on the actual outcrops stripped them of their weathered debris from time to time. In any event, there is ample evidence that only occasional areas of particular beds had any notable amounts of disseminated metals deposited with them, though there is a great vertical range through which beds at many horizons contain them locally. Also, in the Great Basin region the deposits are more generally characteristic of the Triassic in western Utah, as in the Henry Mountain region and similar longitudes in Arizona; but more of them are found in the Jurassic farther east and closer to Pre-Cambrian areas.

The continental territory of the Rocky Mountain region throughout the Triassic and Jurassic sustained a growth of xerophilous vegetation, particularly in southern Colorado and southward and westward, the largest forms being cycads of which there are large fossil trunks. Plant fossils are generally fragmental as if carried down with rock debris by flood streams, though tree stumps are found in place.

FIRST PERIOD OF REDEPOSITION

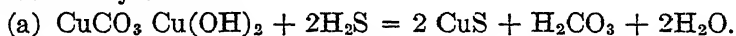
At the close of the Jurassic, the Rocky Mountain area in the region under consideration had been reduced to a peneplain. Across this the Cretaceous seas advanced and buried the metalliferous beds under several thousand feet of sediments. During this long interval much of the fragmental plant material in the beds was converted into coal. Coincident with this, hydrogen sulfide was evolved and retained in the rocks, as were doubtless the volatile hydrocarbons which develop in the formation of coal. Transported iron materials from veins and the iron serving as the red coloring matter of the beds, and some of the copper, probably combined with H_2S as soon as the latter began to evolve, and before the beds were deeply buried, to form pyrite and chalcopyrite.⁴ These minerals are found wholly or partly replaced by chalcocite and covellite, doubtless from sulfate solutions forming in beds which were still in the zone of oxidation. The metal-bearing beds are bleached to a gray color in sharp contrast to the strong coloring of adjacent strata, showing the removal or deoxidation of iron.

³ A. J. Tieje: The Red Beds of the Front Range, etc. *Jnl. Geol.* (1923) **31**, 192.

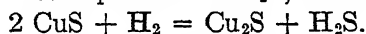
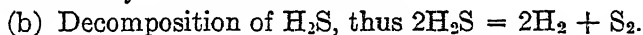
⁴ E. S. Bastin demonstrates the development of H_2S by sulfate-reducing bacteria and suggests the deposition of copper, lead and zinc sulfides from sulfate solutions thereby in the Mississippi Valley. *Jnl. Geol.* (1926) **34**, 773.

Although the disseminated copper minerals were mostly carbonates at the time they were deposited with the sediments, we find the copper redeposited and replacing coal fragments as sulfide, generally as chalcocite. Butler reports silicified plant fragments and notes that these do not contain metals. Such silicification evidently took place before the first period of redeposition. A striking feature of the Jurassic and Triassic beds is the prevalence of lime. They give a vigorous lime reaction, whether they are conglomerates, sandstones or shales. Having then, below water level, hydrogen sulfide and hydrocarbon gases forming in and around the plant remains, and lime, in the presence of copper carbonates, after the beds were deeply buried and temperature comparatively high, the following cycle of reactions can be assumed:⁵

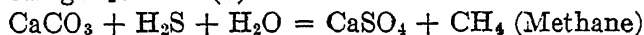
Formation of Covellite



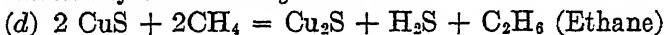
Conversion of Covellite to Chalcocite



Formation of Anhydrite from Calcium Carbonate



Formation of Chalcocite by Methane



Though both are found, chalcocite exceeds covellite, as these reactions lead one to expect. It has been observed that the chalcocite does not always appear as replacements of coal fragments but often in concretionary forms. It is suggested that the latter had their growth about nuclei of lime as indicated in equations (c) and (d). These are not found except in or near the beds containing coal. H_2S is believed to be an essential reagent in both types of replacement. Gypsum occurs as a secondary mineral in the deposits and is accounted for in the above reactions. The solutions were practically stagnant; they apparently deposited the sulfides in the more porous rocks not far from the disseminated minerals from which the solutions were derived and accomplished this more by diffusion than by circulation.

SECOND AND FINAL PERIOD OF REDEPOSITION

In the early Tertiary, the uplift of the present Rocky Mountains began, followed by intense faulting and folding (processes of deformation which are still active) all of which resulted in rapid degradation. Ulti-

⁵ Prof. L. D. Roberts, at the laboratory of the Colorado School of Mines, has found that reactions (a) and (b) are comparatively rapid in distilled water.

mately, and quite recently in some places, the metalliferous beds again appeared at the surface by the removal of the overlying formations, and whenever the beds became exposed the last stage in the growth of the metal deposits began. This may have happened at quite different epochs in the various localities.

In this period, which extends to the present, the deposits have been made over into the form in which they are now generally found at and near the surface, and it is this final stage of concentration which has formed local bodies yielding a small production of high-grade sorted ore. The ores generally referred to in the literature on the subject are those which have been formed in this last concentration, and they are the only ones that approach commercial value in any quantity.

The chalcocite and other sulfides were attacked by weathering agents as the surface was eroded down to them. Such copper and iron as was not carried away in streams was redeposited as carbonates (malachite and azurite), silicates (chrysocolla), and oxides (hematite, limonite, cuprite, etc.). Such chemical redeposition is well known on the outcrops of copper sulfide ores and for a moderate depth below the surface, so requires no explanation. There was doubtless repeated redeposition as erosion cut through such secondary deposits and exposed fresh chalcocitized beds.

The oxidized ores are limited in extent in all the deposits examined and in all those described in the literature. This is due to several causes: (1) The disseminated chalcocite bodies, from which the oxidized ores are derived, were already decidedly localized, as can be seen wherever they have been developed; (2) circulating atmospheric water has free access to them below their outcrops only locally, because the beds are alternating layers and lenses of great variability of texture; (3) structure and topography are usually important factors in the deposition of the oxidized ores; (4) the chalcocite was oxidized and disintegrated and, except at points favorable for redeposition, streams probably carried away considerable copper in sulfate solutions.

Faces of cliffs are particularly favorable points for accumulation. Though the coarse beds were favored in chalcocite deposition, the carbonates seek the finer grained sandy shales. These become saturated with water charged with copper gathered from overlying local chalcocite beds or those immediately below. Such solutions cannot flow freely and drain away, but move slowly outward by capillarity to the outcrops of the shale, where they are evaporated and drop their copper contents. In the bodies inspected, developments showed that the commercial carbonate-silicate ore did not generally extend more than 20 ft. inward from the outcrops along the beds. This seems to be about the distance through which evaporation is effective. The length of such cliff outcrops along the strike, in all cases seen, is less than 50 ft., the carbonate ores

fading abruptly into shales practically unmineralized by copper. In the orebodies, there is special concentration along and near vertical joints and cracks, as would be expected.

Another favorable situation for carbonate ores was found to be on a dip slope where a shale bed lies near and parallel to the surface. Several openings were examined from which high-grade sorted ore had been shipped from such an area. The cupriferous shale was found lying upon a coarse arkose containing scattered chalcocite in nodules and fossil forms with oxidized crusts. Where the arkose was exposed without overlying carbonates, no chalcocite was observed.

Butler stresses the dependence of such deposits upon structure and finds them grouped along anticlines. This may be due to the fact that the beds are usually more deeply eroded along anticlines, cliff faces are formed, and the other conditions noted above are satisfied. He groups the deposits into four classes according to the predominating metal, copper, silver, uranium-vanadium, and manganese, and observes that those characterized by one of the metals is usually deficient in the others. Iron is present in some amount in all these deposits, and at many points in considerable quantities. Manganese locally predominates over iron and a small commercial production has been made in Utah. At certain points in Utah, silver is the principal metal. In Paradox Valley, Colorado, remote from railway, uranium-vanadium beds produced commercially for several years mostly on account of the associated radium, until the radium ores of the Belgian Congo came into competition. Similar deposits near Rifle, Colorado, are now yielding a profit because of favorable transportation. The rarer metals, such as chromium, cobalt and selenium have also been reported, but nowhere in commercial amounts. Copper deposits are, however, the most prevalent and are widely distributed, usually accompanied by a small amount of uranium and vanadium, but in many places free of other metals except iron and manganese. The variations in the contents of the deposits in different regions indicate that particular metals characterized local districts in the Pre-Cambrian ridge.

Butler refers to similar deposits in younger beds, even as late as Tertiary, in the Rocky Mountain region. These were probably derived from older beds and crystalline rocks in the same manner as the metals in the Red Beds, but conditions for accumulation did not prevail so generally.

In the New Mexico deposits examined, vanadium-uranium minerals were recognized at several points, and it is believed that they generally accompany the copper carbonates. Vanadium and its associated metals may have been derived from the plant remains in the beds themselves. It will be recalled that vanadium was discovered in ashes from the burning of coal. An analysis of coal from sandstone beds in southern Colorado

by W. F. Hillebrand showed 0.18 per cent. vanadium. No extended consideration of uranium and vanadium deposits will be undertaken in this paper, but attention is called to the fact that these are found in the same beds as the copper, exhibit the same relations to the other materials of the beds, such as the replacement of carbonaceous fragments, and in their present form are concentrations of oxide minerals, probably derived from minerals originally disseminated.

Lead and zinc were undoubtedly originally disseminated in the beds but were not subject to solution and concentration.

PRACTICAL DEDUCTIONS

The carbonate-silicate ores are not found in sufficient quantity in any one area to justify the erection of milling plants for their treatment. This has been tried at several points in New Mexico with failure as the result.

A larger tonnage might be produced by mining carbonate and chalcocite beds together, but this would greatly reduce the grade if unaltered chalcocite beds were mined, since they are not in themselves commercial. Chalcocite beds may be found enriched to a moderate degree in close proximity to the shale carbonates, but if by mining these a good tonnage of apparently adequate grade could be obtained, such a method would involve the metallurgical problem of recovering copper from carbonates, silicates and sulfides. The sulfides are generally coated and permeated with carbonates and cannot be successfully floated, and table concentration makes a low recovery. The carbonates can be leached successfully, but the silicates only with difficulty. A plant, including table concentration, leaching and smelting of mill products, was tried in the Estey district, Socorro County, New Mexico, and was promptly abandoned. Direct smelting of sorted ore seems to be the only method of making commercial recovery.

No one district known to the writer promises to yield enough production to justify investment in a local smelter. The deposits lend themselves best to the purposes of the individual miner and leaser who can search for lenses of ore, and ship a sorted product to the custom smelters. Under favorable conditions he can make a small profit.

The ores can be found by tracing float in cliff talus or in the wash on dip slopes. Where such guides are lacking the ore probably does not exist. The theory prevailing in some localities that commercial ores are continuous under considerable areas is unsupported by geological evidence. Where the carbonate ores have been found and developed they have been soon exhausted, and the fact that they are discontinuous has been determined in so many cases that it can be accepted as a rule.

DISCUSSION

F. L. Hess, Washington, D. C.—It seems to me that we must treat all of the metalliferous deposits of the plateau sediments together. A number of years ago, I wrote an article⁶ in which I proved, at least to my own satisfaction, that these deposits were all due to precipitation from solution by organic remains, but I never was able to give any formulas for the reactions and I could not discover the original uranium and vanadium minerals. Recently there has been found in a few deposits a jet-black substance, probably a mixture of minerals, which may be the original uranium and vanadium minerals that were precipitated by the organic remains.

Dr. Finch has given an explanation for the copper at the outcrops which is exactly in line with my experience. The same reasoning does not, however, apply to uranium and vanadium deposits; nevertheless it seems to me that the similarity of the causes of precipitation is very close for the three metals.

The silver deposits of Silver Reef, Utah, also are wholly connected with organic remains, the explanation probably being that silver was dissolved by sulfuric and selenic acids formed from weathering minerals in the original deposits and the solutions were carried out into shallow seas from which they were precipitated by decaying vegetation.

D. WHITE, Washington, D. C.—Several points should be noted: First, in relation to the environmental condition of the Red Beds deposition, I should view rather favorably the occurrence of long dry seasons rather than downright year-round aridity.

Second, in general, wherever red-bed series of rocks are found, organic decomposition, in order to bring about the destruction of the plant and animal debris has had to compete successfully against the final complete oxidation of the irons or other metals, with the result that usually these red bed series if they contain much vegetation are not very red, but are apt to be bluish or greenish or light colored. The vegetable debris that we find in red beds is usually drifted.

My third observation is that the buried log is never completely silicified. A part of it, and usually the greater part of it, whether in sandstone or shale, is carbonized, not silicified.

My next point relates to temperature. The Permian Wichita series in Texas (along the Wichita River) carries copper and iron ores which sometimes replace plant material. This series might, perhaps, have been covered as deep as 2000 ft., but the region has never been subjected to any great folding or deformation. The stage of initial metamorphism shows that the beds have not been subjected to any great compressive stresses, such as to cause a temperature of 200° F. In the Appalachian trough, or, better yet, in the Rocky Mountain region at depths of 7000 ft., we find temperatures less than 180° F. Only in regions of active or very recent deformation on the western border of the continent, which have not had time to cool, do we find temperatures approaching 40 or even 30 ft. to the degree Fahrenheit. Exceptions are found near warm intrusives.

In most cases where stems are impregnated with copper it is fairly evident that the silicification—if it is silicification—or impregnation took place before loading of the macerated or saturated material by superimposed sediments had flattened the stems to the collapsed state in which the unmineralized or unimpregnated stems are found.

Finally, again, as to climate: In the northeastern corner of the coal fields of Illinois, at the old mines in Coal No. 2, there are lead and zinc ores which are weathered out of the boney material and the fusain partings of the coal beds. Some may be

⁶ F. L. Hess: Hypothesis for the Origin of the Carnotites of Colorado and Utah. *Econ. Geol.* (1914) 9, 675.

in roof shale. Sometimes there are rather pretty specimens of lead and zinc crystals. In this region embracing the northern part of Illinois there is nothing to indicate any lack of rainfall or failure of the prevalent mildness of temperature when the coal-forming matters were deposited.

MEMBER.—Cobalt has been mentioned. Where is it, and in what form?

F. L. HESS.—The form is very obscure. All I know is that copper, iron and cobalt occur together as oxides in a black-stained sandstone, in Sinbad Valley.

Concerning Dr. White's comments on temperature: these beds must have been buried under at least 7000 ft. of sediment at one time. That might conceivably have brought the temperature up to a full 200° F.

There is another interesting fact about these beds: When they are heated they almost invariably give off organic substances which smell vilely, showing that all the organic compounds have not been broken up, and that the organic remains are not merely in the form of carbon.

D. WHITE.—I do not think any one is assuming that the organic matter in the shales or sandstones referred to is yet reduced to carbon. Apparently the metalliferous salts in solution are reduced or changed to sulfides in the presence of organic matter that is being decomposed and which may furnish the sulfur through oxidation. In its critical moment of hunger for oxygen the organic matter may rob the metalliferous sulfate.

In the Illinois area there has been for a long time very little cover over the coals, and I meant to mention that fact as favoring the migration of solutions. Further, it is very doubtful whether the northern Illinois coal field was ever covered to a depth of 2000 feet.

I wish to stress the fact, which I think you will be able generally to confirm, that where there is replacement of plant stems and branches by sulfides or silica the replacement occurred, almost or quite contemporaneous with deposition, and before the stems had been macerated and flattened under a few hundred feet of superposed strata.

F. L. HESS.—I do not agree with Dr. White about the deposition of most of these things. Before there was any considerable cover on the vegetable remains in the uranium deposits, there was no uranium or vanadium to amount to anything. Deposits of minerals occur where the wood has been partly decayed.

T. S. LOVERING, Washington D. C.—I made a few experiments some years ago that may have bearing on this general question. We have been accustomed to think that organic material will precipitate copper and various other compounds but we have not been able to discover the process by which it is done.

I had some good evidence of bacteria precipitating native copper. We tested out the possibilities and found one or two compounds which had a very strong reducing action; that is to say, we found organic compounds that would reduce copper sulfate to metallic copper rather than to copper oxide. These compounds are all generated by the commoner forms of bacteria, not necessarily bacteria that live only in reducing environments, but bacteria that will live in either a reducing or oxidizing environment. If these small organisms are present, close to the surface, we have a mechanism that is entirely adequate to precipitate copper from rather strong acid solutions.

J. W. FINCH.—Dr. E. S. Bastin⁷ has also touched upon the subject mentioned by Mr. Lovering. I had begun to believe that the reactions described in my paper may have played a part in the deposition of the lead and zinc sulfides of the Mississippi

⁷ See page 390.

Valley. I still believe this to be probable. Dr. Bastin has, however, given much study to the theory of the deposition of these sulfides through the agency of sulfate-reducing bacteria which developed hydrogen sulfide.

E. S. MOORE, Toronto, Ont.—Have you found any reliable evidence that hydrogen is set free in the reaction indicated in the equation given, at the temperature which probably prevailed at the time? I am inclined to question the accuracy of this equation for the conditions specified.

J. W. FINCH.—I do not feel sure about that myself, but at least one metallurgist thinks these processes might have gone on in what he calls a "hydrogen atmosphere." We know, of course, that in the coal fields there is plenty of iron sulfide; we know that in the petroleum we have sulfur, which may have been produced according to one of these equations, with the liberation of free hydrogen.

R. BLANCHARD and P. F. BOSWELL, Duncan, Ariz. (written discussion).—In our scouting we have come more or less in contact with the Red Bed copper deposits of the Southwest. To a lesser extent we have visited the vanadium-uranium deposits. One or both of us have examined most, if not all, of the deposits inferentially designated by Mr. Finch as having been visited by him, and in addition we have examined numerous others, including the Boleo deposit in Lower California, where we spent five weeks.³ Most of the examinations were commercial, but several were made purely for the purpose of increasing our knowledge regarding the deposits. In one case seven weeks were spent in a single district, working out the geology of a large area in which the copper showings exhibited more than the customary promise.

Our observations regarding the Red Bed copper deposits are not wholly in accord with the statements made by Mr. Finch. We concur in his conclusion that most of the deposits, as now known, are not commercial, though Boleo certainly is an exception. We do not concede that some of the deposits, with proper exploration, may not develop into profitable mines, for we do not grant that all the deposits were derived from outcrops now eroded, nor that all of them are necessarily the result of mechanical or chemical concentration by surface agencies. We have failed to arrive at a hypothesis of origin that will cover all the occurrences; for the field evidence shows rather convincingly that different deposits have had different origins.

Commercial considerations preclude our giving as specific details as we would wish regarding certain of the deposits. One of the larger Red Bed areas, in which the copper occurrences are scattered irregularly over approximately 50 square miles, shows no known igneous rocks at the surface within 30 miles; and most of the copper occurs in flat-lying, undisturbed, sandstone beds as replacements of carbonaceous matter by chalcocite, as chalcocite nodules or concretions in the sandstone, and as copper carbonates and silicates replacing the limey cement of the sandstone. But it is noteworthy that if a few of the smaller isolated, copper-stained showings are excluded, the lenses, or pockets, of copper minerals occur entirely within clearly defined fracture or fissure zones of nearly vertical dip; that the largest lenses usually occur in the strongest and best defined fissure zones; that traversing these fissure zones are irregular, discontinuous and generally inconspicuous networks of hypogene quartz veinlets; and that though the quartz has not been thus far found to carry copper sulfides the veinlets are especially numerous where the lenses of copper minerals are likewise largest and most numerous. At one end of the district, isolated from the principal showings, the fissuring over a half-mile area is particularly strong, and this area is flooded with intersecting quartz veinlets that merge into a smaller area of

³ At Boleo the sedimentary beds are not red in color, but in all other respects the deposit conforms to the Red Bed copper type.

garnetized sandstone. Here, not only are the copper minerals associated with the garnetized sandstone; they are intimately intergrown with it.

If space permitted, we could show that the copper lenses above described, with their associated fissure silicifications and strongly defined fissure zones of vertical dip, resemble closely the outcrops over copper orebodies in several sedimentary copper districts of the Southwest. Such showings are just about what would be expected as outcrops over the Bisbee limestone country, for example, if 1000 ft. less of erosion had occurred there. To us, the judging of a Red Bed district such as this one would concern itself not with the possible mining and treatment of the small pockets of mixed copper minerals now showing at the surface, but with (1) whether at depth more favorable horizons existed; (2) if they existed, whether they lay near enough to the surface to make their exploration commercially feasible; and (3) granting (1) and (2), whether the underlying intrusive—which is suggested by the numerous quartz veinlets in the fissure zones, and is clearly indicated by the garnetized sandstone—might lie close enough to the surface to have cut out the more favorable horizons beneath. Judging the deposit from that point of view might lead to conclusions entirely different from those following the judging of the deposit merely on the basis of the scattered small pockets of mixed copper minerals which show at the surface.

In another of the Red Bed districts, which we examined, all the major copper showings in the sandstone (except one that is isolated and is clearly the result of mechanical concentration of desert debris) are either (1) directly associated with an intrusive stock that at the points of contact itself carries copper mineralization, or (2) confined to those portions of the flat-lying sandstone which are cut by numerous nonconspicuous dikes of the intrusive rock. The dikes are not mineralized with copper; the copper in those areas occurs as concretions or as replacements of carbonaceous matter in the Red Beds. But it is significant that in all directions important copper mineralization in the sandstone ceases within $\frac{1}{4}$ mile of the areas cut by the dikes, and that the best copper showings occur in the central portions of the dike areas.

In still another district no intrusive rocks are exposed, but the copper-bearing area, approximately 2 square miles in extent, is domed above the more uniformly dipping sediments, somewhat after the manner of the doming at Ophir, Utah; and the copper mineralization is concentrated closely along two major fissures near the crest of the dome, much as the lead-silver ore at Ophir is concentrated along the mineralized fissures in that district.

We could cite other examples in which the field evidence is persuasive that the present copper bodies in the Red Beds were derived from outcrops now eroded, or that the copper must have been imported mechanically or chemically from an outlying area, as is asserted by Mr. Finch. The three contrasts are mentioned merely to show that a single origin cannot be ascribed to all the Red Bed deposits.

We feel constrained to take exception to the implication that the reason many of the metal-bearing beds are gray in color, as contrasted with the enclosing red strata, is because the copper has deoxidized such beds of their ferric iron. At Bent, for example, in the Tularoso, New Mexico district, the copper of the sandstone occurs in a grayish-white horizon. We doubt if the most optimistic mining promoter, after soberly examining the ground, would assert that one-hundredth part of the exposed horizon is copper-bearing. Yet that sandstone horizon is as devoid of red color $\frac{1}{2}$ mile or more from the copper areas as it is where the copper occurs. The same condition holds for the ore-bearing horizon in the Sierra Nacimiento, near Cuba, N. M. In certain districts we have observed slight local bleaching of the rock immediately surrounding a particular nodule of sulfide or other copper mineral; but when some little copper-bearing lens that could not possibly yield 5 tons of native metal is saddled with

the responsibility of having deoxidized one-half million cubic yards of sandstone of their ferric iron to the extent of changing the color from red to a uniform gray, we incline toward skepticism of the suggested process. When a hypothesis involving so extraordinary a phenomenon is set forth, an impressive mass of field evidence ought to be adduced to substantiate the contention.

It is not our purpose to discuss the various probable origins of the different Red Bed deposits we have examined—we propose first to enlarge our observations so that we may speak with authoritative assurance but we do wish to register a protest against classifying all Red Bed deposits as a unit, and according them, commercially, a blanket condemnation.

R. D. GEORGE, Boulder, Colo. (written discussion).—Dr. Finch has worked out a more satisfactory explanation of the occurrence of copper and other ore minerals in the Red Beds strata than any I have seen. But, regarding equation b, does H_2S break up into hydrogen and sulfur at atmospheric temperatures? I am under the impression that it does so only at a very considerable heat.

I do not know the age of the beds in which the copper carbonates and sulfides (oxides) occur near Carrizo Creek in Baca County, but I am under the impression that they are of Morrison age and do not belong in what we usually class as the Red Beds, though it is highly probable that they may have had a similar mode of origin. The silver deposits of the Lady Belle mine near Brush Creek, south of the town of Eagle, probably belong to the same general category. I am told that the old workings have recently been examined for vanadium and uranium, but with what results I do not know. Certainly vanadium occurs in some of the sandstones associated with the silver ores, but my examination was not carried far enough to determine whether there is a genetic relationship between the silver ores and the rather meager occurrences of vanadium as I found them, but I have been told that outside of the area in which the silver occurred most abundantly vanadium is more plentiful.

J. M. LITTLE, Bloomfield, N. J. (written discussion).—To quote from Mr. Finch: "Since the metals were derived from weathered outcrops, they were transported either as fragments of minerals from the zone of oxidation or as soluble salts, generally as sulfates." ". . . the mineral fragments from copper veins were mostly carbonates."

This theory of syngenetic origin of the Red Bed metal deposits has been advanced before. Professor Lindgren⁹ concluded that the Red Bed copper deposits, at least in New Mexico, are epigenetic; that the ores are disseminations in beds chiefly, by selective replacement, but also in fractures and fissures, and that the fractures and fissures sometimes occur also in overlying and in underlying strata. If this is correct these deposits are not sedimentary metalliferous deposits.

I have examined Red Bed copper deposits in the Copperopolis district in the Zuni Mountains, New Mexico. There the mineralized portions of the beds are apparently where these beds are intersected by mineralized fracture zones, with mineralized seamlets extending below and with the long axis of the deposit along the fracture zone, bearing out the conclusions of Professor Lindgren.

This would, moreover, offer an explanation as to why "there is a great vertical range through which beds at many horizons contain them (metals) locally," to quote again from Mr. Finch.

J. E. SPURR, New Rochelle, N. Y. (written discussion).—I have been much interested in this type of deposit without, as it happens, having had opportunity to study any of the numerous examples. I have read whatever I have found in the

⁹ W. Lindgren, L. C. Graton and C. H. Gordon: Ore Deposits of New Mexico. U. S. Geol. Survey *Prof. Paper* 68 (1910).

literature concerning it, including the discussions of Butler, Ball, and others. Mr. Finch's paper is the most definite presentation of the theory that these deposits are derived through a cycle of changes from materials originally laid down in the beds themselves. While I have been chary of accepting this conclusion, I am quite open to it and much interested in its presentation. Mr. Finch's explanation of the final concentration of the metals is one to which I am predisposed, and I have cited examples of this type of concentration in the last edition of "Geology Applied to Mining."

I am especially struck by the suggestion that the vanadium in deposits of this type may have been derived from the plant remains in the beds. Does the analysis of living plants of any type show the presence of vanadium, which might support that suggestion? I have, of course, been aware that coal and the more volatile hydrocarbons of the albertite type frequently contain vanadium, but I have been inclined to suspect (also, this is of itself a conjecture) that the carbonaceous material has served as a precipitant to vanadium solutions. This whole subject is evidently wide open. All in all this paper is a very marked contribution to the subject.

E. S. BASTIN, Chicago, Ill. (written discussion).—The phenomena of downward sulfide enrichment in most copper deposits present no fundamental chemical difficulties, since primary sulfides are present, which act as agents for the precipitation of secondary sulfides. In the copper deposits of the Red Beds, however, a similar array of sulfides has been deposited in the absence of any older sulfides which can be appealed to as precipitants.

The primary deposition of most sulfide ores of lead and zinc in the western United States is readily explainable through lowering of temperature and pressure in rising solutions of magmatic derivation in which these metals were already in balance with the sulfur radical. In connection with the lead and zinc deposits of the Mississippi Valley region, however—superficial in distribution and presumably also in origin—appeal cannot be made to falling temperature and pressure; another mechanism of precipitation must be found.

Several observers of the copper deposits of the Red Beds and of the Mississippi Valley lead and zinc deposits have sought to explain the precipitation of sulfides as in some way connected with organic matter—coal or bitumen—prevailingly associated with these deposits. However, the fact that the sulfides are often found at considerable distance from any known organic matter indicates that any influence exerted by the latter was very indirect. Whatever direct precipitating agent was operative has left little or no trace of its former presence.

In this dilemma Mr. Finch's appeal to a gaseous precipitating agent such as hydrogen sulfide is particularly alluring not only because it is chemically competent to precipitate most of the observed sulfides but because it is mobile and may travel in solution for some distance from its point of origin before reacting with soluble salts of the metals to precipitate metallic sulfides.

In the case of the Joplin-Miami lead and zinc deposits, as pointed out by both Siebenthal and the present writer, the presence of hydrogen sulfide today in stagnant waters just west of the ore deposits, and even in some of the mines, is strong presumptive evidence that it was at least one of the agencies of sulfide precipitation. In a recent article,¹⁰ I presented evidence to show that the reduction of soluble sulfates through bacterial action offers a competent mechanism to account for the formation of the hydrogen sulfide. Since organic matter in some form is a necessary food for such bacteria, the prevailing association of the ore deposits with organic matter—coal, oil, bitumen, etc.—is understandable, but since the hydrogen sulfide may wander from its source before functioning as a precipitant the spacial association is not necessarily close.

¹⁰ E. S. Bastin: *Op. cit.*

In the Joplin-Miami region sulfate-reducing bacteria are present today in the waters rich in hydrogen sulfide, just west of the mines, and offer partial though not conclusive confirmation of a bacterial hypothesis. As I pointed out in the paper referred to, a verification of this hypothesis in the case of the Red Beds copper deposits would probably be difficult, but it is possible that in connection with some of the deeper deposits ground water carrying hydrogen sulfide may still be present.

W. LINDGREN, Washington, D. C. (written discussion).—I am glad to see that we agree in most part as to the origin of the deposits of the Red Beds. I have always thought that any appeal to magmatic sources is simply out of the question in regard to the deposits.

One of the most interesting parts of the paper was that referring to certain chemical equations, some of which the author says have been verified by Dr. Ward in the laboratory of the Colorado School of Mines. This part deserves further elaboration as to the conditions under which these equations hold good. I suppose that copper carbonate could well be reduced by hydrogen sulfide, and, as a matter of fact, covellite is a fairly frequent mineral in the oxidized zone. I suppose the conversion of calcium carbonate to anhydrite is a reaction that goes on in many places, and I suppose, also, that covellite could be reduced to chalcocite by CH_4 , but I think a little further discussion of these equations would not be amiss. I have a theory that the presence of uranium and vanadium as well as that of chromium and selenium is related to organic reactions, the details of which have never been worked out. It is also an interesting fact that the chalcocite nodules occurring in the Red Beds of Oklahoma contain quite a little silver.

W. H. EMMONS, Minneapolis, Minn. (written discussion).—At the present time, when geologic opinion trends strongly toward igneous sources to account for most deposits of the valuable metals, it is especially interesting to read a paper that presents a group of carefully worked out hypotheses to account for the origin of the widespread copper ores in the Red Beds of the Southwest. These ores are so different in composition and environment from the normal quartz veins and similar ores with clearly marked magmatic affiliation that one is very naturally disposed to ascribe to them a different origin. The whole group of deposits, including the copper ores of the Red Beds type, and the uranium and vanadium ores of the Southwest, are so far from igneous sources and so nearly independent of ores of other common metals that no other hypothesis appears probable.

The author ascribes the precipitation of the copper sulfides to the reaction of the copper-bearing solutions with hydrogen sulfides and suggests that the latter may have formed through the agency of bacteria, a process recently investigated by Dr. Bastin. In 1905, I examined certain deposits of this group, yielding copper and silver, situated on the south side of Paradox Valley, Colorado, and east of the La Salle Mountains. In this region on the north side of the La Salle group there is a spring of cold water from which hydrogen sulfide now issues rather copiously. The copper ores and the hydrogen sulfide spring are remote from known outcrops of igneous rocks and do not seem to have any close igneous affiliations.

The hypothesis of Dr. Finch that vanadium ores may have originated from organic sources suggests an attractive problem for investigation. When I first read Spencer's descriptions of the unusual lead and zinc phosphates and vanadium minerals that were found with bones of animals and the relics of early man at Broken Hill, Rhodesia, it seemed probable that the secondary vanadium minerals might have formed like the secondary phosphates, through the reactions of metal-bearing waters with the material dissolved from bones. Frank Notestein, who was investigating the geology and chemistry of the vanadium compounds, made analyses of bones to ascertain

whether they contained vanadium, but in the limited material available to us at that time he obtained only negative results. It is possible that the vanadium is derived only from vegetable matter, as suggested by Dr. Finch.

The source of the chromium in the vanadium deposits near Placerville, Colo., described by Hess,¹¹ presents a difficult problem. Chromium is one of the least mobile of the metals in hot magmatic solutions. Neither is it very mobile in ground waters, although it does form chromates of metals in certain lead deposits in serpentine. The investigation of the chemistry of the copper, vanadium and chromium ores and their close associates in the deposits of the Red Beds group would probably throw much light on the general problem of the genesis of mineral deposits.

¹¹ F. L. Hess: *Op. cit.*

Geologic Factors in the Development of the Eastern Pennsylvania Slate Belt*

BY CHARLES H. BEHRE, JR.,† CINCINNATI, OHIO

(New York Meeting, February, 1928)

THIS paper deals with recent geologic studies in the slate belt of Northampton, Lehigh and Berks counties, Pennsylvania. The work was conducted under the auspices of the Pennsylvania Topographic and Geologic Survey. Three years have now been spent in detailed consideration of the economic geology of this region. In the field, particular attention was paid to the bearing of geology on quarry development and property valuation and the object of this paper is to summarize the results from that viewpoint. Excellent studies of a somewhat similar nature have already been carried out by Bowles,¹ but with the emphasis upon the details of quarrying and fabricating methods, rather than upon the geologic relations.

THE NORTHAMPTON-LEHIGH-BERKS DISTRICT

Location

The region here discussed lies in east-central Pennsylvania and extends southwestward from Delaware River, on the New Jersey line, to the Schuylkill in north-central Berks County. It is an elongated belt occupying the northern sections of the three counties mentioned and lying along the southern edge of Blue Mountain. Its length is about 50 miles and its average width five or six, though in places it attains a maximum of 10 miles.

There are no large cities in the region, but about 10 miles south are Reading, Allentown, Bethlehem and Easton. Railroads serving the slate belt are the Delaware, Lackawanna and Western, and the Pennsylvania along the Delaware River, the Lehigh Valley and Central of New Jersey along the Lehigh, and the Pennsylvania and Philadelphia and Reading on the Schuylkill. In addition, two railroads—the Philadelphia and Reading and the Lehigh and New England—cut obliquely through the slate region to reach the more isolated communities. Transportation facilities are thus excellent.

The slate deposits of these three counties are conspicuously unified, both economically and geologically. In general type of marketable

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¹ Oliver Bowles: *The Technology of Slate*. U. S. Bur. Mines *Bull.* 218 (1922) 1.

products, in the age and attitude of the slate-bearing rocks, in composition and appearance of the slate, in quarrying and fabricating methods, and in the interlocking of the capital responsible for the development of the region, the slate deposits in the three counties may well be regarded as a single district. From the geological viewpoint this is especially true, for a single formation furnishes all the slate and beds are virtually continuous across the region.

Importance

In the production of slate Pennsylvania ranks first among the states of the Union. In 1926, the value of slate² produced in this state was \$5,413,177, about 26 per cent. greater than that of Vermont, its nearest competitor. No other state had a production value in excess of \$1,000,000. Of the total quarried and sold in Pennsylvania, less than half a million dollars' worth came from outside Berks, Lehigh and Northampton counties. Indeed, in 1927 there was only one other producing district in the state and but a single operating quarry in that district; the slate from this quarry is crushed for granules and flour. It may truly be said, therefore, that the Northampton-Lehigh-Berks region is by far the most important in this country.

Slate comes on the market in various forms, the major classes being roofing slate, structural slate (slabs sawed and dressed to a greater or less degree, to be used as walls in showers and toilets, as grave vaults or as billiard table tops), blackboards and bulletin boards, slate employed for electrical insulation and school slate. Large quantities are also crushed and ground into granules or flour, for use in making composition roofing, paint, rubber and the like. The colored slates of Vermont are in special demand for roofing, though also largely utilized for electrical purposes. For the most part, Maine slate goes to electrical uses. The Virginia and New York output is mainly for roofing. Pennsylvania produces roofing, structural and electrical slate, as well as almost the entire blackboard and school slate supply in the United States. The great variety of products of the district here considered is thus obvious. In 1926, a normal year, there were in the region 43 producers operating about 60 quarries.

GENERAL GEOLOGY OF THE DISTRICT

Stratigraphy

The geologic map of the three counties (Fig. 1) readily explains the great linear extent of the slate belt.

The productive area is underlain by the Martinsburg formation of upper Ordovician age. The slate belt stops to the south against the

² For these and succeeding figures see A. T. Coons: Slate in 1926. U. S. Bur. Mines, *Mineral Resources of the U. S.* (1927).

earlier Ordovician limestones, one of which is the cement rock quarried at Bath, Northampton, Egypt, Fogelsville and elsewhere. These older rocks are brought to the surface by a northward monoclinial dip, the regularity of which is locally concealed by close folding and much faulting, chiefly of the thrust type. To the north, the slate is covered by the Silurian Medina conglomerate. These broad relationships have been known as early as the middle of the last century.³ The more detailed geologic mapping has been carried on recently by B. L. Miller⁴ and the writer,⁵ who have described the various structural details.

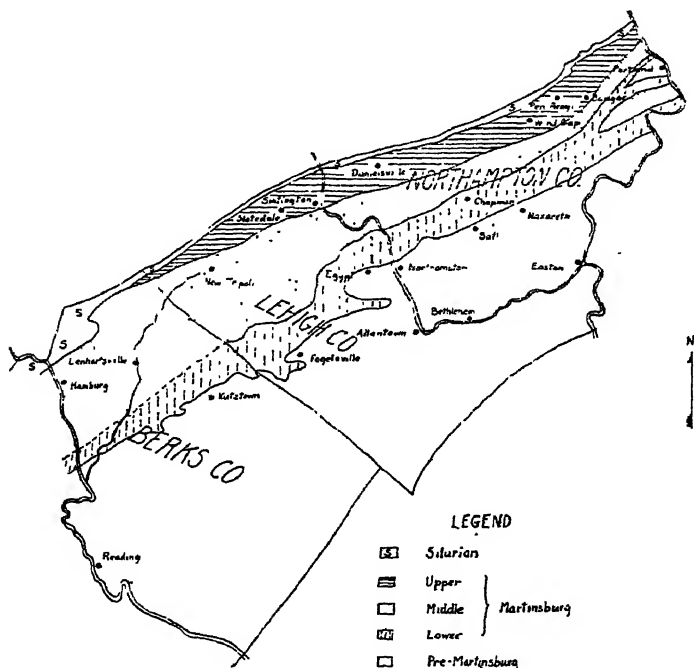


FIG. 1.—GEOLOGICAL SKETCH MAP OF THE SLATE BELT OF NORTHAMPTON, LEHIGH AND BERKS COUNTIES, IN EASTERN PENNSYLVANIA.

The slate rock itself may be divided into three parts—an upper member, which is a slate with relatively few beds of calcareous sandstone, a middle member consisting largely of sandy sediments, and a lower member that differs in composition from the upper mainly in containing numerous thin siliceous bands.

It is in the upper member that the greatest development has taken place. The rock of this part of the Martinsburg consists of alternate blue-gray (micaceous), dark gray or virtually black (carbonaceous) and

³ J. P. Lesley: Summary Description of the Geology of Pennsylvania. Pa. 2nd Survey (1892) 1, 543.

⁴ B. L. Miller: Allentown Quadrangle. Pa. Geol. Surv. Atlas, No. 206 (1924).

⁵ C. H. Behre, Jr.: *Jnl. Geol.* (1926) 34, 481. Pa. Geol. Surv. Rept. M.-9 (1927).

light gray, buff-weathering (sandy) beds. As these alternating layers are generally not in excess of a foot in thickness, large blocks of slate that are free from banding or "ribbons" are not common and care must be exercised to obtain "ribbon"-free slates. For most purposes, clear stock is at a premium but "ribbioned" slate (Fig. 2) can be used for structural purposes and for roofing.

The lower member yields a good grade of slate, but here the beds are so thin (seldom more than 3 in. thick) that it is impossible to obtain "unribbioned" slate. Moreover, the flat cleavage surface of this slate is generally not absolutely smooth, the cleavage plane rising slightly as it crosses the more siliceous beds; this does not weaken the slate as a roofing material, but it differentiates it slightly from the typical slate of the



FIG. 2.—RIBBONED ROOFING SLATE FROM THE "SOFT" SLATE BELT OF NORTHAMPTON COUNTY. CLEAR SLATE IS PREFERRED.

upper member. As the slate of the lower part of the Martinsburg formation contains more siliceous matter than most of the upper, it is frequently spoken of as "hard" slate, to distinguish it from the more micaceous or "soft" slate of the upper member. These terms, it should be understood, refer to hardness upon abrasion, not to resistance in weathering. In 1927, only three quarries were being worked in the lower member.

The middle Martinsburg, although largely sandy, does in a few localities bear a little slate that has been worked with profit. Its most valuable product, however, is a red and green shale, generally classed as slate, which is quarried and crushed or ground for paint, filler or roofing.

The Martinsburg formation, in common with the older Ordovician and Cambrian rocks, suffered folding toward the close of Ordovician time; it was then eroded unevenly, in such a way that the upper member thins westward, after which the Medina conglomerate, which forms the crest of

Blue Mountain, was laid down. The effect of the erosion interval between the two formations has been to cause the upper member to disappear westward from Quaker City on the Berks County-Lehigh County line. This is shown in Fig. 1.

General Structure

The general structure of the region is a monocline, striking northeast and dipping northwest. In detail, however, there is intricate folding along northeast axes, and some faulting. Although the faulting is locally important, it is the folding particularly that concerns the quarry operator. Toward the eastern end of the region the beds have suffered close compression and northward overturning, the axial planes of the folds being almost horizontal; but westward from Wind Gap, the folds stand more nearly vertical and the axial planes dip south at steep angles.

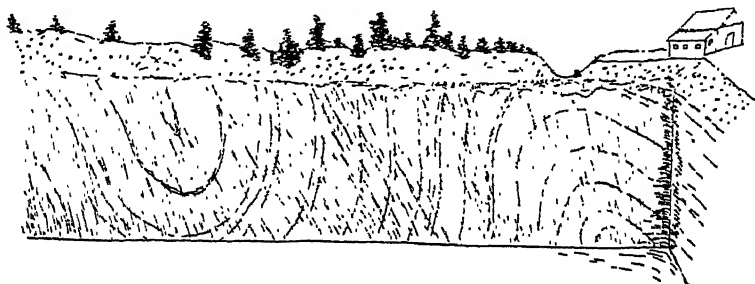


FIG. 3.—TYPICAL FOLDING OF THE SLATE BEDS OF THE UPPER MARTINSBURG, AS ILLUSTRATED IN THE EAST WALL OF THE EUREKA QUARRY.

Cleavage shows as oblique lines from upper left to lower right. The beds are thrown into a southerly anticline and a northerly syncline. Note dip of axial planes of folds to south. Compare with Fig. 8.

This is accompanied by a change in the attitude of the cleavage. A typical fold of the region is shown in Fig. 3.

The cleavage that is found in slate is the result of pressure. Whether it forms at right angles to the direction of maximum compression⁶ or in the planes of maximum shear⁷ is as yet undecided, but observers are agreed that cleavage develops along planes that bisect the axes of the folds into which the slate beds are thrown. Consequently the cleavage of the slate in the eastern section is close to the horizontal; in the western section it is more nearly vertical, standing at angles of 50° or more. This in turn causes some differences in the quarry methods of the two sections, which will be reverted to later.

⁶ C. K. Leith: *Rock Cleavage*. U. S. Geol. Surv. *Bull.* 239 (1905) 109.

⁷ G. A. Becker: *Experiments on Schistosity and Slaty Cleavage*. U. S. Geol. Surv. *Bull.* 241 (1904) 11.

The above statements apply chiefly to the "soft" slate. The "hard" slate shows almost flat cleavage planes wherever opened.

In addition to bedding and cleavage, grain and jointing may be mentioned as other structural features of importance in quarrying. Grain is in a direction of easy parting approximately at right angles to cleavage; it is due to the orientation of the greatest elongation of mineral particles having three different dimensions. It generally has a strike of N. 45 W. throughout the region.⁸

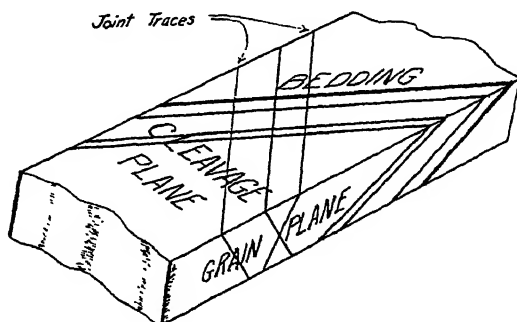


FIG. 4.—STRUCTURES IN A BLOCK OF SLATE AS QUARRIED.

Note that cleavage and grain planes are approximately at right angles. Bedding and joints may intersect other structures at any angles, but more commonly strike parallel to cleavage.

Jointing, although not as constant in direction as cleavage or grain, is commonly marked by strikes of approximately N. 60 E. In certain places where faulting has occurred, jointing is very closely spaced and spoils large tonnages of slate.

Fig. 4 illustrates these features, showing their appearance in a slate block as quarried.

APPLICATION OF GEOLOGY TO QUARRYING

Correlation of Beds

Only those who actually operate quarries in the "soft" belt of Lehigh and Northampton counties realize what a significant role is played by stratigraphy. The beds of the "soft" belt are divided into groups or "runs," each of which is marked by one or several conspicuously thick beds. Each such thicker bed is readily distinguishable by its peculiar properties, such as color, behavior on splitting, and electric conductivity. As the thicker beds yield clear stock in large slabs, it is preferable to open the quarries if possible in them, and the portions of the sequence in which thinner beds alternate are regarded, at worst, as waste. Exposures other than those of the quarries, however, are rare and so sections have to

⁸ C. H. Behre, Jr.: Slates of Northampton County, Pa. *Jnl. Geol.* (1926) 34, 503.

be pieced together with care. Correlations are established by measuring sequences of beds, noting intervals between exceptionally thick beds and details of thick and thin, light and dark bands, and by recognizing certain beds that possess unusual hardness or softness or special mineral or chemical composition. Correlation thus is not unlike that used in determining

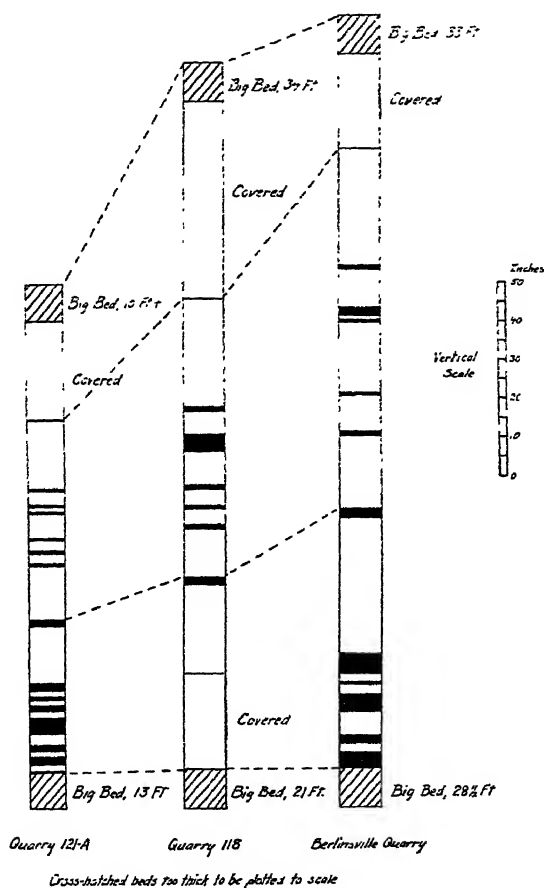


FIG. 5.—MEASUREMENTS OF BEDS IN THREE QUARRIES, TO ILLUSTRATE CORRELATION METHODS.

Distance between Quarry 121-A and Berlinsville Quarry is 3 miles along the strike.

the structure of coal. Fig. 5 is a sample comparative chart showing the closeness of agreement between measurements of the same beds.

Certain beds are very much preferred to others. Most of the light gray slate is sericitic, but chloritic beds do occur and these generally sell at a premium over other slate. Such chloritic rock has a faintly greenish cast, and splits and wears exceptionally well. Examples are the Gray bed of the Albion and Gray Bed "runs" and the gray bed of the Danielsville quarries (which probably corresponds to the Albion Gray bed).

Other slate beds are very dark and thus especially suitable for the making of school slate. The commonness of such nearly black beds in the quarries at Slatington has led to the building of a large school-slate industry near that city. Again, other things equal, the thickest beds are preferred. Thus the upper and lower Washington big beds (which are respectively 33 and 25 ft. thick at Slatington) and the Franklin (two big beds, the upper of which is 13, the lower 16 ft. thick at Slatington), though both yielding generally excellent slate, are especially valuable because of the large proportion of usable slate in the total quantity quarried. From the thick beds all slate produced is "ribbon"-less; hence the rough block may be trimmed with an eye for the greatest sizes and the minimum of waste. Such exceptionally thick beds are sought along their outcrop; if they are not readily accessible at the surface, tunnels are driven from vertical shafts to tap the desired big beds and to mine them as far as the lateral limits of the property.

The colored slates found in Berks County in the middle member of the Martinsburg present a special problem. These are beds of red and green clay-slates in which the cleavage is not as well developed as in the gray slates of the region. They have been quarried at Lenhartsville and Greenawald, and formerly at Albany as well. At present a steady though small production is maintained at the two first-named localities. The brilliant colors are due chiefly to variations in the content of ferrous and ferric iron in a chloritic and sericitic clay slate. The beds are continuous along the strike and are apparently thin, lenticular deposits of origin similar to the gray slates. Though closely folded, such individual lenses can be traced along the strike for distances of four miles. Attempts have been made to work them for roofing slate, but without success.

Slates of this type are known not only within the area here described, but westward as well. They were studied by the writer as far west as in the Swatara valley, midway between Hummelstown and Annville. Their widespread occurrence makes their development dependent more upon closeness to transportation than upon any other factor.

Effect of General Structure on Quarrying and Mining

If the beds are nearly horizontal, a large opening is likely to be made. In such a case, to work a valuable bed over its full extent, operations are carried on first by stripping until the desired bed or beds are reached, and then by quarrying laterally down the dip. At Bangor, where the beds lie flat, stripping and subsequent large-scale quarrying are the methods in common use, and here the quarries opened are of exceptional size; examples measure 1000 by 500 ft. (Consolidated Star Quarry) and 1200 by 500 ft. (Old Bangor opening). At Pen Argyll similarly large openings are made, but here the size is due mainly to the great length

of the quarries along the strike of the beds, the developing companies having bought up sufficient land in the early operating days to enable considerable lateral extension without infringing upon adjacent properties. As the beds at Pen Argyl stand nearly vertical, great depths are attained, the deeper quarries reaching 600 ft. At Slatington, on the other hand, the beds are also nearly vertical, but show repeated

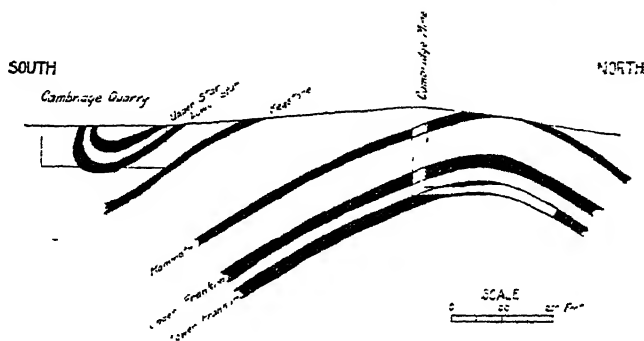


FIG. 6.—SECTION THROUGH CAMBRIDGE MINE, SLATINGTON, ILLUSTRATING METHOD OF FOLLOWING A BED DOWN DIP BY SLOPE MINING.

turns at shallow depths; further, the development is intensive so that only smaller holdings are available along the strike of the more valuable beds and this has the effect of cutting down the strike length of openings. At the same time the valuable beds near Slatington are thicker and hence more sought after and there is a general prejudice against taking out the thinly banded slate above and below the thick beds; hence relatively small openings are made at the surface, resembling large

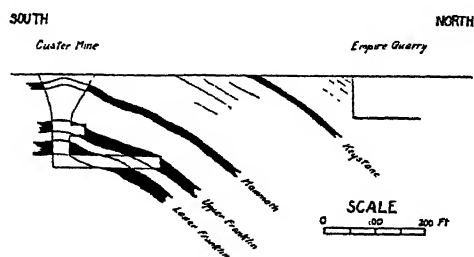


FIG. 7.—SECTION THROUGH CUSTER MINE, LITTLE RUN JUNCTION, ILLUSTRATING SHAFT APPROACH IN MINING BEDS THAT DO NOT OUTCROP.

shafts rather than quarries, and frequently underground mining is actually carried on down the dip of the beds after exhaustion of that part of a stratum which can be removed by ordinary quarry methods. Sketch sections of such mining operations studied in 1927 are shown in Figs. 6 and 7, respectively.

At Slatington and Slatedale, many of the quarries practice a sort of underground mining intermediate between quarrying and room and

pillar mining. Where the beds have inclinations of 40° or so, they are followed on the slope, the roof consisting of the more closely ribboned slate which is not sufficiently valuable to justify removal. Large beds especially are worked in this way.

The advantage of underground mining in place of quarrying is obvious. There is little or no ice in the winter, the temperature is equable, and the men can continue to work in wet weather. On the other hand, no effort is made to ventilate, with the result that water vapor, escaping steam and smoke unite to confuse the sight. In addition, the roof must be carefully watched, as bedding planes are not infrequently parting planes and slides may well become dangerous.

The Cambridge mine at Slatington, with about 300 linear feet of underground workings, is at present the most extensive operation of this type. This mine is electrically lighted. Its production supplies one selling company. Not far from it is the New York tunnel, a large mine, in which the accessible passageways alone total some 600 linear feet; much slate was taken out here, the chambers being opened on slight dips parallel to the beds. This mine is now abandoned.

Effect of Pitch on Property Values

Were the folding in the region wholly around axes having little or no pitch, the structural problems, though complicated by irregular topography, would still be relatively simple. But the folds all possess unmis-

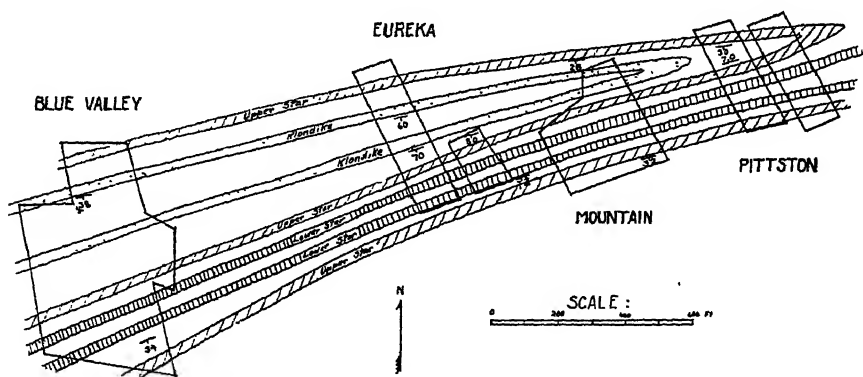


FIG. 8.—SKETCH MAP SHOWING GEOLOGY AT QUARRIES NEAR EUREKA.

Three big beds—the Klondike (above), Upper Star, and Lower Star (below)—appear in a syncline and anticline. Axial planes of the folds dip south. Compare with Fig. 3, which serves as a section across the strike of the structure.

takable pitch and it is rarely indeed that there is parallelism between cleavage and bedding strike, as should exist with horizontal fold axes.

The values of adjacent properties may differ widely as a result of the effect of pitch in removing beds. In places the inclination of the axis of a fold is so slight as to be not obvious on inspection, but else-

where it is sufficient to cause the disappearance of a bed on one side of a property line, which on the other side, a hundred feet distant, was well developed. The map and cross-section at Eureka afford an example (Fig. 8). The Klondike big bed, which was quarried in the Blue Valley and Eureka openings, was completely lacking in the quarries farther east because of the westward pitch of the syncline. This disappearance of the bed took place in a horizontal distance of less than 300 ft. The bed in question has a thickness of 24 ft. at the fold axis; the distance separating the east wall of the Mountain quarry (where the Klondike bed is last seen 25 ft. below the surface in the trough of the syncline) and the west wall of the Pittston quarry (where the beds beneath the Klondike appear at the surface) is 240 ft.; the pitch of the fold axis must therefore be at least 12° and probably is more nearly 15° or 20° .

Effect of Overburden

A part of Eastern Pennsylvania has been repeatedly invaded by the Pleistocene glaciers, with the result that the slate is frequently covered with an overburden that makes quarrying expensive and dangerous. At Slatington, the East End quarry was sunk through 35 ft. of glacial drift and several quarries northwest of the town encountered till and

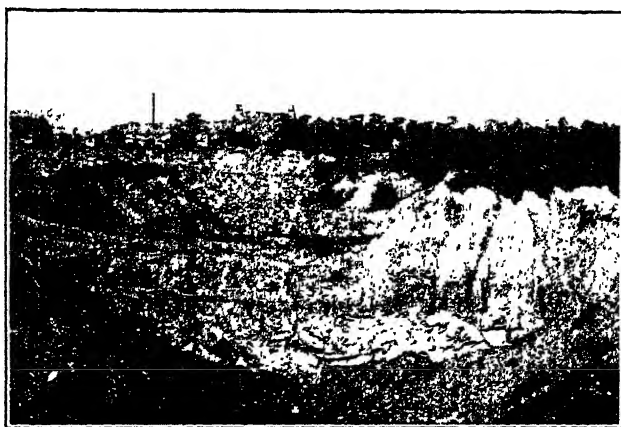


FIG. 9.—OVERBURDEN AT THE BLUE RIDGE QUARRIES, SLATINGTON.

Glacial drift 30 ft. thick has been stripped well back from the quarry holes, two of which show in the foreground and middle distance. Cable supports and boiler house may be seen in far distance.

water-laid material that necessitated careful cribbing of the opening. Fig. 9 shows the heavy overburden at some of the Slatington quarries, partly stripped back so as to lessen the danger of a slide into the hole. This condition holds even more generally from Pen Argyl eastward, for Wisconsin glaciation affected the eastern part of Northampton County, leaving a heavy till cover. From the general neighborhood of Lynn-

port westward the glacial overburden is slight and slate is encountered only a short distance below the surface. Quarry owners are at times able to avoid the cost of stripping newly developed property by entering into agreements with highway contractors to furnish them with fill or road basing.

Effect of Jointing on Quarrying

This has not been adequately considered by the average owner and operator. The feeling apparently is that joints follow no definite systems and cannot be predicted. In 1923 the writer studied two quarries in both of which the arrangement of joints was manifestly radial and upon this basis he recommended that quarrying be abandoned. The company operating one quarry subsequently failed and the other quarry, after changing hands, is being worked at a loss. Both openings were located in the troughs of folds whose axes lay at right angles to the monoclinical structure of the region; that is, these minor folds pitched northwest. It is apparent that transverse strains will be set up in such structures and these may be expected to find relief in joints whose patterns will only conform to the general joint patterns of the slate on the edges of the structure. Where two or more joints converge, V-shaped blocks will break out and the three corners of such blocks will all have to be discarded in trimming the slate to block size; the quarrying of such triangular blocks is therefore very uneconomical.

Jointing of a different type is also noticeable along fault planes. In several quarries, as they were sunk downward, zones were encountered along which movement nearly parallel to the bedding and apparently subsequent to the development of the cleavage had taken place. The Bangor Superior and Albion quarries at Pen Argyl illustrate this type of fault. It has also been observed in the Peach Bottom slate district in southern Pennsylvania and the writer believes it to be more common than is generally recognized. Slate quarrymen frequently speak of such faulting as a "loose ribbon," but this term has been applied to other structures having a wholly different origin. When the movement took place the friction between the two sides opened numerous inconspicuous cracks which now break the slate into such small blocks that quarrying is not profitable for distances of 20 or 30 ft. on either side of the fault plane.

Effect of Relation between Cleavage and Bedding

A low opening angle between cleavage and bedding plane makes it possible to get a longer cleavage surface free from ribbons, as is shown in Fig. 10. For this reason, operations at Pen Argyl and Bangor can with profit be carried on in beds which are only half as great in actual thickness as those at Slatington, the cleavage and bedding intersecting at low

angles; this is especially true of the quarries in the neighborhood of Bangor.

In the middle member of the Martinsburg, sandy and slaty beds alternate in close succession, so close, indeed, that the slaty beds are frequently less than an inch thick. The quarrying of cleavage blocks would under ordinary circumstances yield slate that was crossed by ribbons. Locally, however, the bedding meets the cleavage at very low

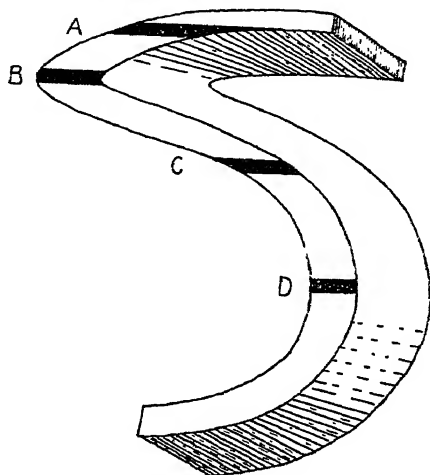


FIG. 10.—STEREOGRAM OF A SINGLE SLATE BED, SHOWING CLEAVAGE-BEDDING RELATIONS.

The cleavage will bisect the opening angles of the folds. Note that at *A* a longer piece of slate is obtainable along the cleavage, even though the bed is here actually thinner than at *B*, *C* or *D*.

angles and hence long slabs of slate may be obtained that show no ribbons. At Rockdale and near Scheidy small quarries were opened where this condition exists. Unfortunately, the slate here possesses a strong tendency to split along cleavage as well as on bedding planes, so that the pieces obtained are triangular in vertical section, thinning out along the convergence of bedding and cleavage. Quarrying should not be attempted under these unfavorable conditions until such a time in the distant future when the more easily worked and higher grade rock has been exhausted.

Effect of Close Folding on Thickness of Beds

Although there is a general thickening of beds westward, this primary effect due to variations in deposition is not the only factor affecting the beds. Where compression has been intense, the slate has been squeezed so as to thicken and thin with regard to its position on the fold. This matter has been discussed elsewhere⁹ but it may be summarized by saying that, as would be expected, there is a pronounced thickening on the crests

⁹ C. H. Behre, Jr.: *Op. cit.*, 490.

and in the troughs of anticlines and synclines and a corresponding thinning on the limbs, and also that in those places where differences between the south and north limbs of synclines are noticeable, there is a marked relative thinning on the south limb.

The decrease on the south limbs of synclines was first noticed at Bangor. At Slatington and Slatedale, toward the western edge of the "soft" belt, the difference between the south and north limbs is far more marked. Here the quarrymen speak of the south limb of synclines as the "back pitch." The beds on such a structure show a remarkable thinning on the south limb of the troughs and as this affects primarily the slaty beds, the sandy beds remaining more nearly constant in thickness, the south limbs are practically not worked. Near Lynnport two quarries have been opened in the south limbs of synclines but without success.

Effect of Cleavage Attitude on Quarry Methods

There is a noteworthy difference between the attitude of cleavage in the eastern and western parts of the district. In the east, at Bangor and Pen Argyl, the cleavage is more nearly horizontal, the average dip in 20 quarries selected at random being 18° . At Slatington the dip is far steeper; in 20 quarries from this region the average is 56° .

The effect of this is to change quarry methods. At Bangor and Pen Argyl the flat cleavage surfaces admit of the use of channelling machines; not only is it not complicated to run the channeller on a track, but the blow is downward and gravity assists it. In the Slatington region, however, the channeller can only be called into use in rather exceptional locations. Here it is not uncommonly employed in the sumps of the openings or in taking out the key block. But very much of the work is done by drilling, because the drill can more easily be mounted so as to drive horizontally. Parenthetically it may be added that considerably more powder is used here than at Pen Argyl and Bangor.

Relation of "Curl" to Pitch

In many places the slate shows a feature called "curl" by the quarrymen. This consists of cracks or very small discontinuous joints that cut the cleavage surfaces of the slate along lines the traces of which are generally more or less at right angles to the cleavage dip. Microscopically examined, these fracture planes are seen to be due to the crinkling and breaking of cleavage planes, as Dale¹⁰, and earlier writers have pointed out. They were therefore developed subsequently to that reorientation of the mineral particles which is the cause of cleavage, and their presence indicates the occurrence of a period of renewed strain in the slate.

¹⁰ T. Nelson Dale and others: *Slate in the United States*. U. S. Geol. Surv. *Bull.* 586 (1914) 37.

Certain conditions are exceptionally favorable to the development of "curl:"

1. Where thin beds of slate alternate with beds of sandstone, folding induces drag of the slate by the more rigid sandstone; the slate, unlike the shale of which it was originally composed, yields to strain of this type as a nonplastic medium and the cleavage is wrinkled and finally broken.

2. If strong folding takes place, after the slate has developed cleavage, the slate is broken as indicated, even though there be no drag by more rigid layers. In such cases the strikes of the "curl" planes and of the cleavage are parallel.

3. In the development of pitching troughs, (or crests) there is considerable movement between the beds. In the axis of the trough (or crest) the strain caused by this differential movement is parallel to the grain or cleavage and the stress is relieved to a large extent along these directions by the mineral elasticity of the slate; but anywhere else in the fold the movement is oblique to grain and cleavage (which are, respectively, at right angles and parallel to the axial plane of the fold); this oblique stress expresses itself by wrinkling the cleavage at an angle to the cleavage strike.

The latter cause is especially important and doubtless accounts for much of the "curl" in quarries like the Custer and East End at Slatington. "Curl" is especially common in the region from Danielsville west. Not all slate affected by it is worthless, but care must be taken in the use of "curly" slate for a roofing slate affected by it may split after laying in such a way as to intersect the nail holes and it then may fall from the roof.

GENERAL IMPROVEMENTS IN QUARRY PRACTICE

As a result of geologic field studies, four recommendations looking toward a greater return on slate quarrying may be made.

Core Drilling

Core drilling is the only method by which the slate operator, no matter how experienced, may assure himself of the favorableness of stratigraphic and structural conditions before opening. Drilling should be carefully carried out, the cost being regarded as a legitimate expenditure comparable to development work in a mining operation. In the Slatington district, where close folding makes the underground structure irregular and where it is desired especially to open only certain exceptionally thick beds, it is more generally practiced than elsewhere in the slate region (Fig. 11). As recently pointed out by Bowles,¹¹ it may

¹¹ O. Bowles: Recent Progress in Slate Technology. U. S. Bur. Mines *Rept. of Investigations* No. 2766 (1926) 6.

serve another purpose, for drilled holes sometimes facilitate the drainage of quarries.

In the Slatington district, shot drilling has been the form most generally resorted to. One or two among the experienced slate operators have from time to time contracted to drill exploratory holes on land of which the development was contemplated, thus carrying on a very useful type of consulting work.

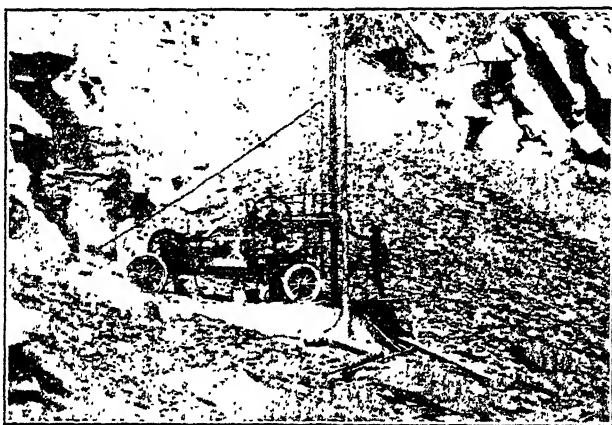


FIG. 11.—EXPLORATION FOR A BIG BED BY CORE DRILLING, SLATINGTON.

The writer has found that, thanks to the constancy in their direction, grain and cleavage combined may be used to roughly orient drill cores if they become turned in pulling. In the district here considered, cleavage is known generally to dip south and grain to trend about N. 45 W. Given these two directions, therefore, the attitude of the bedding can readily be determined in any given core.

Scientific Quarry Valuation

Very little has been done in the slate quarries, either in Pennsylvania or elsewhere in the United States, in developing methods for quarry valuation. In advance of development, valuation contains a strongly speculative element, but after the opening has been made, certain reasonable assumptions are admissible for postulating definite quantities of available slate. Such estimates should consider the following factors:

1. Total quantity of clear stock available, computed on the basis of a reasonable working depth.
2. Total quantity of ribboned stock available, on the same assumption.
3. Loss due to waste, taking into account irregularities in cleavage and jointing.

4. Loss incidental to operation, taking into account method of opening (*i. e.*, whether by drilling, loosening with powder, or channelling) and waste in trimming blocks to proper size.

5. Special factors enhancing the value, such as the use of certain beds for electrical or blackboard purposes.

Cost Accounting

No quarries can be successfully operated over long periods of time without accurate cost-accounting methods. These are especially essential in fixing the price at which the slate is to be sold. Business management thus becomes a necessary perquisite to successful quarrying. Not only does cost accounting give the final dependable criterion upon which to base the price of the product, but it is often the deciding factor in determining which of two alternative quarry methods is to be followed.

Conservatism in Opening New Quarries

On the whole, there is a tendency toward conservative development in the slate district. But anyone well acquainted with the industry must realize that it is at present more nearly over than underdeveloped. For that reason, caution is advisable. The extensive opening of new

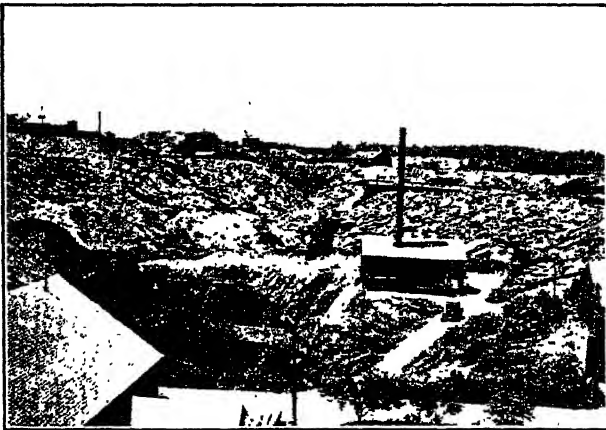


FIG. 12.—QUARRIES AT SLATEDALE, LEHIGH COUNTY.

Of five openings shown here in a row, only two are now being worked. Note the piles of waste slate.

properties is to be discouraged rather than otherwise. It is believed that quarries in the "hard" belt, which yields only roofing slate and very limited classes of structural slate, will generally operate at a disadvantage in comparison with those of the "soft" belt, because there is so little opportunity to utilize material not suitable for roofing. It

is true that the capital expenditure is less in "hard" belt operations, because the "soft" belt mill must be equipped with trimming, planing and dressing machinery; but these costs are at least equalled, if not overbalanced by the sale of mill products. Those sections in the "soft" belt where operations have been unsuccessful in the past, due to relatively poor slate, might better be left untouched until such time as the more readily quarried and more advantageously sold material is nearer exhaustion. The situation in slate quarrying recapitulates that in copper and iron mining, in which, until relatively recently, only the higher grade deposits have been worked; as supplies go down and demand up, not only will the better grades of slate bring a higher price, but the insistence of a large class of consumers upon a cheaper product will permit the saving of much material of a sort that at present is discarded as waste (Fig. 12) because too closely "ribboned." The analogy would apply yet more strictly were it not true that there is no dearth whatever of slate and that the exhaustion of the higher grade material is not even faintly in sight.

DISCUSSION

G. R. MANSFIELD, Washington, D. C.—To what extent has the operation of the slate mine been controlled in the past by geologic work? Has any quarry actually been laid out on the basis of geologic work?

C. H. BEHRE, JR.—The Amalgamated Slate Corp'n. is doing that for the first time. There has been indirect control always, because the slate men themselves are quite canny; they get on a certain bed and stay on that bed as best they can. Frequently that is done by guess. There has, however, in the last 15 years, been a great deal of core drilling among the slate men. This drilling is carried on with considerable care and frequently a quarry operation is abandoned simply because it is known, after core drilling, that the operators cannot get the bed they want. Usually it is the older hands who have "learned by hard knocks" who refuse to accept the use of geology; the younger ones are gradually coming to recognize its value.

O. BOWLES, Washington, D. C.—Professor Behre has worked for several years on the study of geology in the slate district and his work is outstanding in its practical application. It is the only real geological work of which the practical application has been emphasized, but we all know that it is difficult to make the operators profit by this information. It is a process of slow education.

D. FOCHT, Easton, Pa. (written discussion).—Mr. Behre says, quite truly, that there is considerable difference between quarry methods of the Bangor-Pen Argyl region (east) and the Slatington region (west) on account of the large difference in the angle of cleavage. In the Bangor-Pen Argyl district the track channeler and more recently the wire saw (which was introduced in 1926, and has been installed in several of the Pen Argyl quarries during the past year) are found to be the most satisfactory devices for cutting the quarry. In the Slatington region, however, drilling and broaching methods are used to the best advantage.

The cleavage planes of the Bangor-Pen Argyl region dip at gentle angles in most quarries, and therefore allow the track channeler to be set up without much bracing; the machine, which moves on a track under its own power when cutting, can easily

move back and forth along the cut without an elaborate system of counterweights, or some other external source of power, which would be necessary to pull the carriage up a steeply pitching cleavage plane. The blow is of the hammer type, and, as Mr. Behre points out, gravity assists in the almost directly downward blow. The track channeler under very good conditions will cut about 7 sq. ft. per hour. An average cut is 18 ft. long by 8 ft. deep. Its main objections are the slow speed of cutting, the expense to operate and maintain, and the damage it does to rock adjacent to the cut.

The track channeler is being quickly supplanted by the wire saw, but never will be entirely supplanted; in the best equipped quarries, use may still be found for the channeler for short cuts.

The wire saw is also very adaptable to the flat cleavage. Here, core holes 36 in. in diameter have to be drilled at either end of the proposed cut. A shot core-drilling outfit especially developed by the Ingersoll-Rand Co. has proved best suited for this purpose. These core holes may be set 100 ft. apart (longer cuts have been made with varying success). Posts or standards anchored in these holes are equipped with a guide pulley, which may be lowered to the bottom of the hole by means of a screw. An endless three-strand wire passes through the pulleys, which are gradually lowered as the cut progresses.

Sand and water are fed to the cut. A cut 100 ft. long and 10 ft. deep has been finished in 48 hr. after the outfit is set up, for an average cutting speed of about 20 sq. ft. per hour. The wire saw has the added advantage of not destroying any rock. However, it has limiting factors in its use. The core drill, as it is at present, cannot cut at an angle greater than 30° from the vertical. The saw must also cut at an angle less than the horizontal angle, and therefore cannot cut in a horizontal plane, on account of the feed of sand and water which is necessary to cutting. With the present core drill, horizontal holes cannot be put in. The machine is not required to meet these difficulties in the Bangor-Pen Argyl region, but in the Slatington region it would be called upon to meet them.

At Slatington we have, as Mr. Behre says, a cleavage angle of about 50° against an average of 20° for the region to the east. Besides this, the nature of the bedding has caused a different operation altogether from that followed at Pen Argyl, where the quarries resemble large shafts with more or less vertical sides. In the Slatington region, however, tunnels or mines are the general practice, which follow the various big beds characteristic of the region. At the Genuine Washington quarry, at Berlinsville, we have a typical example of the operations in this district. Here we quarry slate from the two Washington beds which lie about 18 ft. apart, narrow ribbon beds intervening, which we remove with the main beds. These beds run horizontally for a short distance and then pitch downward at a gradual angle. In order to remove the slate in solid blocks, the roof and one side of the quarry (the east) should be cut. This is necessary also as a measure of safety in order to have a good solid roof for the tunnel. For this cutting we have found that drilling and broaching give the best results. In the first place, the quarry bar upon which we mount the drills is light and easily moved to the difficult positions which are necessitated by the steeply dipping cleavage. The drill giving the best service at present is of the reciprocating piston type; it is manufactured by the Sullivan Machinery Co. and classed as the Hyspeed FL3 rock drill. With this drill we can put in 27 holes 8 ft. deep in 15 hr.; the broaching requires an additional 9 hr., which gives us a cutting speed of about 4 sq. ft. per hour. This is very slow, but the portability of the drill makes it faster than any other practical method.

We use a 1¾-in. hollow drill steel sharpened with an X bit 3¼-in. dia. for a starter. A set of drills comprises three lengths. The broachers are 4 by 5⁄8 in. with a cutting edge of 4½ by 1½ in. containing 10 teeth, and conforming to the drills in length.

The four tunnels at this quarry require three drills in constant operation and at times it is necessary to put on a fourth in order to keep ahead with the cutting.

The use of the track channeler is out of the question. In the first place, it is a heavy machine, and could not be moved around to the difficult positions and set-ups necessary without false work and auxiliary hoists. These would interfere with the removal of blocks from the tunnels, as they would be in the way. At the angles at which the cutter head would have to operate, the cutting would probably be slower than with the present method.

The wire saw, of course, is faster, but the core drill is also a heavy machine and could not be operated on the steep cleavage planes, as the maximum angle at which it can be operated is 30° from the vertical. The saw itself would hardly operate on a horizontal cut, as the feed of water and sand would not be effective. Nevertheless, under conditions where the bedding is dipping downward sufficiently to allow the core drill to operate and the cutting effect of the sand to be at a maximum, the wire saw could be used with much profit. In the Slatington region, where the conditions are lacking, the wire saw in its present stage of development cannot well be used.

Clay Prospecting and Mining in California

BY W. F. DIETRICH,* STANFORD UNIVERSITY, CALIF.

(Los Angeles Meeting, September, 1928)

THIS paper deals with the methods of mining the high-grade clays of California. The fact that the majority of the clay pits in the state are operated on a scale that is small by comparison with most metal mines causes a diversity of methods and gives rise to problems of selection of the method best suited to given conditions. Two or three operators in California produce approximately 500 tons per day of a single kind of clay from a single pit. About five varieties of clay are mined on a scale of 200 to 400 tons per day; not over 15 varieties are mined on a scale of 50 to 200 tons per day; and more than 50 varieties are mined at the rate of less than 50 tons per day.

It is the aim of this paper to delimit, as far as the data at hand permit, the conditions and scale of operations that favor the selection of each dominant method of mining that is actually being applied at California properties.

Details of the occurrence and utilization of the clays of California have been given in a recent bulletin¹ and will not be considered here.

GEOGRAPHICAL AND GEOLOGICAL FEATURES OF CALIFORNIA CLAY DEPOSITS

The greater part of the high-grade clays consumed by clay-working plants in California is mined from one of three producing areas in the state: (1) Lincoln, Placer County, on the eastern edge of the Sacramento valley, 30 miles northeast of Sacramento; (2) Ione, Amador County, in the foothills of the Sierra Nevada, 38 miles northeast of Stockton, and 12 miles west of Jackson, one of the important towns of the famous Mother Lode gold-mining district; and (3) Alberhill, Riverside County, 90 miles southeast of Los Angeles.

Other districts of minor importance are near Cardiff and Carlsbad, San Diego County, and El Toro and San Juan Capistrano, Orange County. In addition, there are a number of isolated properties in various parts of the state.

* Associate Professor of Mining Engineering, Stanford University, Calif.

¹ W. F. Dietrich: The Clay Resources and the Ceramic Industry of California. Cal. State Div. of Mines and Mining, *Bull.* 99 (1928).

Lincoln District

The clays of the Lincoln district occur in flat-lying beds aggregating over 100 ft. in thickness. These are continuous and are exceptionally uniform in quality over areas of several hundred acres, and the overburden is less than 12 ft. thick over most of the district. At present, these clays are being exploited on a scale of several hundred tons per day by each of three different companies, one of which, Gladding, McBean Co., has a large manufacturing plant on a site adjoining the clay pits. The Clay Corporation of California, a subsidiary of the Stockton Fire Brick Co., produces clay for the parent company's plants and sells clay to other manufacturers. The Lincoln Clay Products Co. is an independent producer, having no plants of its own.

At all three properties, the occurrence of clay in large and uniform beds and the scale of production are such that power-shovel mining has been adopted. At the property of the Lincoln Clay Products Co., stripping is removed with a power shovel, but the clay itself is now being mined by hand.

The clays from the Lincoln district are used mainly in the manufacture of architectural terra cotta, firebrick, faience tile, and stoneware. Broadly considered, they may be classed as buff-burning refractory bond clays. The total production from the district is about 100,000 to 120,000 tons each year.

Ione District

The clays of this district occur in deposits of Eocene age scattered through a belt about 10 miles long and 2 miles wide. Starkweather and Bacon are the important operators in the district. They carry on a general clay-mining business from pits owned or leased by themselves, or act as contract miners for other properties. Two of the important firebrick manufacturers operate their own mines in the district. There are several smaller independent operators.

There are about 16 active pits or underground mines in the district, from some of which more than one variety of clay is mined. The production from each pit is relatively small, and only four of them are consistently mined on a scale in excess of 50 tons per day during the working season. These larger properties are mined by power shovels. The smaller pits are mined by hand or scraper methods, unless the overburden is too thick, in which case underground methods are employed. At one or two of the pits, a power scraper is used, and the clay is loaded into auto trucks through the customary trap door arrangement. At the smaller pits, the equipment is meager, and in general the clay is loaded by hand shoveling into small auto trucks, for transport to the nearest railroad siding.

The Ione clays are mainly refractory sands containing from 15 to 25 per cent. of fireclay. There are also a number of important varieties of plastic fireclay of exceptional quality, and some face-brick clays. A clay-washing plant has recently been installed and produces a washed clay of good quality. The total production of sand and clay from the district is over 75,000 tons per year, about half of which is sold independently by Starkweather and Bacon.

Alberhill District

The Alberhill clays occur in an area in the Temescal Valley about 12 miles long and 2 miles wide. The general trend of the valley is northwest-southeast. The town of Corona is at the northern extremity of the workable deposits, and Alberhill is near the southern end. Clays are found on the side slopes of the adjacent mountain ranges, especially that on the east, as well as on the valley floor itself. The clays are of Eocene age.

Clay-mining conditions in the district are characterized by the necessity of mining specified tonnages of each of over 30 different varieties of clay, from deposits that are in general irregular in shape and size and lack homogeneity. On the other hand, each of the operating companies produces a sufficiently large tonnage to warrant the maintenance of adequate equipment for all needs; and furthermore, the operations of each company are centralized within a radius of not over one mile from the manufacturing plant or railroad loading bins to which the clay is delivered. Thus the problems of transfer of material, of transportation and of supervision are greatly simplified, as compared to the conditions that are typical of the Ione district.

The extreme irregularity of the individual beds, and their variations of dip, render power-shovel mining impracticable at some properties, but at the Emsco pit, between Corona and Alberhill, and at some of the pits on the floor of the valley at Alberhill, conditions are favorable for the use of small gasoline-driven shovels.

Most of the deposits, with the exceptions noted, are mined by hand, in open pits, glory holes, or by underground methods. In the open pits, the clay may be loaded directly by hand into motor trucks, or into wheelbarrows dumped into loading chutes from which trucks are loaded.

The Alberhill Coal & Clay Co., the largest independent operator in the district, has found it necessary to use underground methods to a greater extent than have the other operators, mainly because it is prepared to supply any reasonable quantities of each of some 32 different varieties of clay, as called for by customers. As a consequence of the rough topographic environment and the irregularity of the company's deposits, it has been necessary to follow some of the beds beneath the surface. Glory-hole mining has been wisely employed at a number of

places on the property. Room-and-pillar mining is also used, and wherever possible the clay is broken down into chutes from which it can be withdrawn by gravity.²

The clays of the Alberhill district include many white, buff and red-burning varieties, of use in the manufacture of firebrick, face brick, terra cotta, stoneware, earthenware and other products. The refractory and face-brick clays are the most important. The total production from the district probably exceeds 100,000 tons per year. Reliable statistics are lacking.

Cardiff-Carlsbad District

This area contains numerous varieties of refractory and face-brick clays of Eocene and Pleistocene age. Individual deposits are scattered over a considerable area, and practically all are owned or leased by manufacturing companies of the Los Angeles district. One of the properties is mined at the rate of 150 tons or more per day, and a power shovel is used. The output from other properties is less, and hand or scraper methods are used. The annual production of high-grade clay from the district is about 30,000 tons, all of which is consumed in the Los Angeles district.

Southern Orange County

Near El Toro, and at other points in southeastern Orange County, are important deposits of fireclay of exceptional quality. The known deposits are controlled by various manufacturers of clay products of Los Angeles, but some varieties are obtainable on the market. Both surface and underground mining methods are used. The total output of high-grade clay from this part of Orange County is now close to 20,000 tons per year, mainly from two separate deposits.

PROSPECTING

Up to the last few years, comparatively little systematic prospecting of the clay deposits of California has been needed. The clays outcropped on the surface, and a superficial examination, with the aid of a few test pits or open cuts, served to demonstrate the existence of a deposit of sufficient size and uniformity to justify commercial operations. With the growth of the industry, the consolidation of smaller companies into larger units, and the impending exhaustion of some of the more easily workable deposits, there has been a rush on the part of many of the larger companies to acquire clay lands for future use. This has given rise to more careful prospecting than was required in the past, and in all

² For a detailed description of the property as of 1923, see J. H. Hill: Clay Deposits of the Alberhill Coal & Clay Co. California State Mineralogist's Rept. (1923) 19, 185.

of the principal districts of the state, systematic prospecting by drill holes or by test pits has been employed.

Where the maximum depth to be drilled does not exceed 30 ft., and where the physical nature of the formations permits, hand augers are favored. In the Lone district, in places where the overburden is not too indurated and is free from large rock fragments, one man can drill 30 to 50 ft. of hole per day, to average depths of 20 ft. Under such conditions, the extremely low cost of drilling greatly favors the method by comparison with open-cut or test-pit work, unless it is desired to secure large samples for testing.

The preferred type of prospecting drill for average depths in excess of 30 ft. and in hard formations is a portable gasoline-driven core drill, using a toothed cutting head of steel. These rigs can be easily hauled from hole to hole by a light tractor or a team, and are strong enough to drill holes up to 4 in. dia. to depths of 200 ft. in the formations usually encountered at clay properties. The cost of such work is about \$1 per foot of hole.

Where a capping of volcanic rock or other indurated material is present, and where the holes must be drilled to average depths of 100 to 300 ft., a heavier rig, involving a combination of rotary and churn drilling, is convenient.

Whatever the method of drilling, the disposition of the samples must be carefully considered. It should be obvious that the expense of prospect drilling is such that the operator is not justified in losing the opportunity of securing all pertinent information that may be available; yet, in a few instances, it has been noted that samples have been carelessly sacked and labeled, and stored unsystematically without making adequate notes in the drill logs, or attempting to accurately classify and map the kind of clay disclosed. Many drill logs merely state that the clay is "blue," "yellow," or "white," in color, and is "sandy," "plastic," "hard," or "soft," in structure or texture. Of course, such reports are found only where the operators have been interested in finding one particular kind of material and have deliberately neglected other formations, yet it would seem that ample provision should always be made to secure complete records of all formations penetrated by the drill, so that attempts may be made to utilize all materials that overlie or immediately underlie the principal commercial bed. The least that can be done is to keep accurate depth logs of all holes, and to save suitable portions of all cores obtained, filing these so that they can be referred to readily.

In a number of investigations of clay land undertaken in recent years in California, very complete geological, engineering, and ceramic data have been taken, and the reports of those in charge have been as thorough and complete as the reports of the best metal-mine examinations. In the Lincoln district, the most important criteria of the value of a given

variety of clay are chemical analysis and softening temperature (pyrometric cone equivalent); hence these determinations are made on all core samples. The results, when plotted in plan and section, serve to delimit the boundaries of the various commercial varieties of clay.

At Alberhill, on account of the numerous varieties of commercial clay, an adequate prospecting campaign must provide for complete preliminary clay tests.³ Here the important criteria are shrinkage, fired color, fired porosity, softening temperature and plasticity. Chemical analysis is of relatively little value. The fired data need not be taken at many different temperatures if the clays are to be used in plants belonging to the company sponsoring the prospecting work, but should be done at the temperatures typical of the various products for which the clay might be adapted. Clay producers with a general market would be justified in having more complete tests made, so that they may classify each variety of clay more accurately for the benefit of customers who desire clays with specifically defined properties. In all cases, the laboratory test data on the most important varieties of clay should be followed by semicommercial tests under the conditions prevailing in plants where the clay might find use.

Due to the expense of the tests outlined, there is a tendency to rely upon visual and tactual observations. These are warranted for the preliminary classification of all of the less important varieties of clay, or for remote parts of the mining area. But the core samples should always be systematically preserved, so that they may be available for further investigation.

OPEN-PIT MINING METHODS

The open-pit mining methods in use in California clay mines consist of various combinations of the following methods of loosening, loading, transportation and dumping:

Loosening:	None
	Hand picking
	Hand auger drilling, followed by blasting
	Plow
Loading:	Hand shoveling
	Scraper: Horse-drawn
	Tractor-drawn
	Drag-line
	Power shovel
Pit transportation:	None. Loaded directly into auto trucks or cars
	Wheelbarrows
	Mine or industrial cars
	Auto trucks
	(Scrapers)

³ For standard testing methods, see *Jnl. Am. Cer. Soc.* (1928) **11**, 335-534; particularly 448 *et seq.*

Loading for transportation to plant or railroad:	None. Hauled direct from pit to plant or railroad Hand shoveled from platform Gravity delivery from chute or bin Belt conveyor delivery from hopper
Transportation to plant or railroad:	Same vehicle as in pit Auto trucks Industrial railroad: Hand tramming Horse tramming Locomotive haulage Rope haulage .

Loosening

Drilling and blasting are necessary at most pits mined by hand methods, but are seldom required where the clay is mined by power shovels. When drilling is required, hand augers are in general use. The holes are usually drilled into the toes of the banks, rather than from the tops of the benches. Where drilling is required as a systematic part of the mining method, at pits where hand mining or scraper work is used, the holes are placed at intervals of 2.5 to 3 ft. and are usually from 4 to 6 ft. deep. One man can drill and blast about 100 ft. of hole per shift. Blasting at most pits is done with low-grade dynamite, usually No. 2 stumping powder, using cap and fuse firing.

Plows are used at only a few pits, where clay or overburden lies in a nearly flat, comparatively thin layer, too tough to be mined without loosening. More use is found for the plow in common clay pits than at the high-grade pits. For outputs over 50 tons per day, it is generally economical to use light tractors, rather than horses, to draw the plows.

Loading

Hand shoveling is used at all pits where the output is small and irregular, where hand sorting or selective mining is required, where the deposit is thin, or where other conditions mitigate against mechanical methods. In general, outputs less than 30 tons per day, deposits less than 4 ft. thick, working seasons of less than three months, most cases of irregular geologic structure, and all cases requiring hand sorting, favor hand mining.

Scrapers, usually of the Fresno type, are used in mining flat deposits that have sufficient vertical homogeneity to permit the mining of successive thin horizontal layers without disturbing the uniformity of the product, and where the daily and seasonal output are large enough to warrant the purchase and maintenance of equipment. Generally, if more than 30 tons and less than 100 tons are to be mined each day, scraper methods will be applied if other conditions are favorable. Horse-drawn scrapers are little used, as the operating cost of a small tractor is less than

for the equivalent animal power, and maintenance costs are low. While conditions usually favor the tractor-drawn scraper rather than the drag-line type, the latter is popular at many pits, especially where common clay or shale is being mined. At one common-brick plant, a clamshell excavator is used.

Numerous power shovels are in use. These are all of the revolving type and have a dipper capacity of $\frac{5}{8}$ to $1\frac{1}{4}$ cu. yd. A caterpillar-type gasoline shovel of $\frac{3}{4}$ yd. dipper capacity costs \$6000 to \$9000, and can excavate up to 500 tons per shift, if operated continuously. Gasoline (distillate) is the preferred fuel, but steam (oil-fired) shovels are in use at a few pits, and an electric shovel has been applied in at least one locality. Although, in one instance, a power shovel is used for a daily output of but 50 tons, the minimum output that will, in general, justify the purchase and operation of a shovel is nearer to 150 tons per day, for a working season of at least four months, and an expected life of at least five years. Where a second-hand shovel can be obtained at a bargain, or where one shovel can be used at two or more adjacent pits during the season, these limitations can obviously be more liberal. If a shovel is on hand, its use is justified for outputs in excess of 50 to 75 tons per day. For all outputs under 150 tons per day, no pitman or fireman is ordinarily needed for the operation of the small shovels in general use.

The Hogger shovel, a simple device mounted on a Fordson tractor, is being given a trial by one company, and the outcome is being awaited with interest. This shovel can be purchased for \$1400, and under favorable conditions it will load 150 tons per shift.

Pit Transportation

The selection of the method of pit transportation depends primarily on the method of loosening and loading; also, the distance of the pit from the plant or from a standard-gage railroad connection. Where the clay is to be transported by rail to the point of use, it is sometimes possible to use spur tracks on the pit floor, which will permit direct loading of cars at the pit face.

In many cases, auto trucks of various sizes are loaded directly on the pit floor, by any of the loading methods already discussed. Many different arrangements are found in practice, some of which indicate questionable engineering judgment. For example, at one pit a tractor-drawn scraper was in use, for delivering the clay into a small hopper from which a 1-ton truck could be loaded. The pit crew consisted of three men, exclusive of the truck driver. The hopper held barely more than one truck load, and the trip to the railroad siding was 5 miles each way. Although there was some stripping to be done, and occasional drilling and blasting, the pit crew was idle for 50 to 65 per cent. of the working

clay. The daily output, at best, was 15 tons, at a total cost, delivered to the railroad, of approximately \$2.50 per ton, exclusive of stripping. This was a case where the pit was equipped for producing several times more clay per day than was required. It would have been possible to remove the stripping at the beginning of the season by contract to local owners of scraper equipment. The clay could then have been mined by hand shovel and wheelbarrow work, dumping into a 1-ton or 2-ton loading hopper from which the truck could have been filled. The wages of one man and the cost of operating the gasoline tractor-scraper would have been eliminated, making the unit cost approximately \$2 per ton, exclusive of stripping. If the desired output exceeded 30 tons per day, the scraper system would be preferred, as the number and size of trucks could be increased to the point where the scraper would be in practically continuous operation.

At a number of pits, light trucks have proved economical for direct transportation from pit face to a central dumping point, in preference to mine or industrial railroads. Two examples of what would seem to be sound practice follow. At one pit, from which about 25 tons per day were being mined, a 1-ton truck was loaded by hand shoveling at the face, which was less than 100 yd. from a railroad siding. Three men were employed, including the truck driver, and there was no more idle time than was necessary for rest periods. At a Southern California pit, several miles from the railroad loading point, and at a considerable elevation above the adjacent valley floor, a gasoline shovel was used to load 1-ton trucks, which delivered the clay to a loading chute at the mouth of the pit, about $\frac{1}{4}$ mile from the face. From this chute, larger trucks were loaded by gravity for the haul to the railroad. The daily output of 200 tons required three men in the pit crew, and two trucks, a total of five men, to deliver the clay to the transfer chute. The capacity could have been doubled whenever necessary by adding more trucks. A foreman supervised all operations, including the trucking and shipping. Charging half his wages to the pit, the direct labor cost was about 14 c. per ton for mining. Allowing for shovel and truck operation and maintenance, the total mining cost did not exceed 30 c. per ton, delivered to the transfer chute.

Wheelbarrow work finds its customary place in the clay pits for small-scale work, for delivering clay from the face to loading chutes or platforms not more than 150 or 200 ft. from the working face. For greater distances, or for large outputs, other methods of pit transportation are preferable, such as mine cars or industrial railroads. A great variety of methods are in use, including hand tramming of small mine cars, horse tramming of industrial cars, gasoline or electric locomotive haulage, and wire-rope haulage. No great divergences were noted from the practices familiar to metal-mine operators.

Loading Arrangements

At some pits, the clay is loaded at the face into railroad cars or plant bins, but in most instances the pit haulage system is separate from the auto truck or railroad that is used to transport the clay to the point of use. In such cases, suitable transfer arrangements are needed.

There is a tendency to slight the importance of suitable storage capacity, with the result that pit operations must be closely synchronized with the transportation system, or with plant requirements. At some pits, there is no storage whatsoever between the pit and the main transportation system. It would seem that in most cases less time would be lost in the pits, and the transportation system would be more efficiently operated, if the pit equipment delivered the clay to a storage bin at least large enough to hold a single transportation unit. In many cases noted, the pit was idle, except for minor activities, whenever a railroad car or auto truck was not available into which the clay could be loaded. It is true that the cost of a bin is not always justified, especially if a bucket elevator or belt conveyor must be used to transfer the clay from the ground level to the top of the bin, but careful consideration will usually permit the selection of a site upon which a relatively cheap bin with suitable approaches can be built. This is particularly true in those districts where the prevailing topography is hilly, as at Lone and Alberhill.

In addition to the lack of storage capacity, another possible source of inefficiency at a number of pits is the use of a flat loading platform instead of a gravity chute for loading railroad cars. During the summer months most of the loading is done by dumping the trucks or wheelbarrows directly into gondola cars, from a platform having a hinged apron which is dropped inside the car. During winter months, and at other times when box-car transport is desired, box cars are loaded by hand shoveling and wheelbarrow work from flat platforms at the level of the car doors. For winter storage, large open-sided sheds enclose the loading platforms. The storage pile is built up by the trucks, with the aid of hand spreading, without mechanical conveying systems. On the open side along the railroad track, a dry wall is built up of lump clay to retain the storage pile. While this arrangement permits ample storage capacity at moderate expense, it involves an additional handling cost, by comparison with a gravity loading system. The present (August, 1928) contract price for hand loading box cars from a platform is 20 to 25 c. per ton. Since the cars are filled only to a height of 40 in., leaving a clearance of nearly 4 ft. to the tops of the doors, it should be possible to devise a simple and efficient portable conveyor system, which would save at least 10 c. per ton over present costs. Only one such device was noted in practice. This consists of two portable conveyors, one of which is placed inside the box car and receives its load from the other conveyor, which is fed by shovel and wheelbarrow work. The entire apparatus cost less than

\$1000. Operating costs are not available, but a comparison of labor cost may be made from the fact that a five-man crew drills, blasts and loads 100 tons (2 cars) of clay in three days.

SURFACE VERSUS UNDERGROUND MINING

The criteria of choice between surface and underground mining in the clay pits of California have not been definitely established. In most cases, the decision has been made without the benefit of accurate cost comparisons, and without considering the conservation of clay for the future. As a result, wide variations of practice are to be found. There is a tendency, however, to use glory-hole or underground methods wherever the thickness of the capping exceeds the thickness of the beds to be mined. This would indicate that the cost of underground mining, after due consideration of the value of clay left in pillars, is approximately twice that of surface mining, per ton of material handled.

In the Ione district it is generally believed that underground mining of sand beds 12 to 16 ft. thick should be used whenever the thickness of the capping exceeds 8 to 10 ft. Royalty leases at a flat rate per ton for a period of time shorter than that required to mine out the easily accessible clay have been largely responsible for this condition. A contributing factor has been the intensive competition for market, with resultant low selling prices for clay and sand. The result has been that no operator could make a profit unless he almost totally disregarded the future. On surface operations, the permanent damage from following this policy is slight. At the worst, future costs will be but slightly increased, owing to the necessity for rehandling stripping that was dumped on clay land, the releaning of abandoned faces, and the irregularity of pit faces left by present-day gophering practices; but there will be no great loss of clay, whereas in the underground mines practically all of the clay left in the pillars, amounting to 25 to 40 per cent. of the original deposit, will be absolutely lost by admixture with capping.

Criticism of the operators who have been responsible for this condition in the past is not warranted. Clay was so plentiful and so easily recoverable that no profit could be made unless selling prices were reduced to the lowest possible figure. It is probable that the end of this phase is at hand. In the Ione district, and elsewhere in California, it is unlikely that the discovery of new sources of supply will more than compensate for the annual tonnages mined. The more important known resources are now controlled by large organizations, able to give the clay-mining problem careful consideration with the view of obtaining the maximum net profit from the property. Although the principles that apply to this problem are well known to mining engineers, they are so pertinent to the future of clay mining in California that a numerical example does not seem out of place here.

Let it be required to find the thickness of capping that can be stripped economically from a clay (or sand) deposit 16 ft. thick. The deposit contains 200,000 tons of clay that might be recoverable by surface methods. The annual production of clay is to be 10,000 tons, at the rate of 100 tons per day, and a shovel with a capacity of $\frac{3}{4}$ cu. yd. is to be used. The cost of removing overburden and providing dump areas is 30 c. per ton. The cost of mining and delivering to the pit mouth by shovel, in conjunction with a suitable method of haulage, is 30 c. per ton. The cost of mining by underground methods, with the aid of a mechanical loading machine, is 45 c. per ton, delivered at the pit mouth. Miscellaneous costs by either method, including 10 c. per ton royalty, amount to 65 c. per ton delivered into railroad cars. With underground methods, 30 per cent. of the clay must be left in pillars. The selling price of the clay is assumed to be \$1.25 per ton.

Under these conditions, the life of the property is 20 years if mined by open-pit methods, or 14 years if mined by underground methods. The net operating profit by open-pit methods is

$$\$1.25 - (\$0.30 + \$0.65) = \$0.30 \text{ per ton, exclusive of stripping.}$$

By underground methods, total costs are \$1.10 per ton, leaving \$0.15 per ton as net operating profit. At 8 per cent. interest, the present value of recoverable clay is \$29,454 if mined by surface methods, or \$12,366 if mined by underground methods. The difference between these, or \$17,088, is available for removing stripping, if the surface method is used. This is equivalent to an annuity of \$1740 for 20 years, which, at 30 c. per ton, would pay for the removal of 116,000 tons of capping, or 58 per cent. of the clay available. In other words, if the capping is $9\frac{1}{4}$ ft. thick over a deposit 16 ft. thick, the two methods would yield an equal net profit. There are, of course, certain refinements which may be introduced into these calculations whenever the basic data are sufficiently reliable or the margin of difference small enough to warrant them.

It is obvious that the economic ratio of capping to clay is increased by (1) greater net operating profit by both methods; (2) greater cost differential in favor of the open-pit method; (3) lower mining extraction by the underground method; (4) shorter life of property without increasing amortization costs, and (5) lower assumed rate of interest. Taking the foregoing problem as an example, in which it was found that the limiting ratio was 0.58 tons of capping per ton of clay, and varying one factor at a time, leaving all others as in the original problem, the effect of each variable is indicated in the following:

1. If the selling price were \$1.75 per ton, 0.85 tons of capping could be mined per ton of clay.

2. If the net operating profit by the underground method were \$0.10 per ton, the limiting ratio would be 0.72.

3. If the underground mining extraction were 60 per cent., the ratio would be 0.62.

4. If the life of the property were 10 years by open-pit mining (and 7 years by underground mining), the ratio would be 0.61.

5. If the assumed rate of interest were 6 per cent., the ratio would be 0.60.

GLORY-HOLE MINING

The glory-hole method has been extensively used at Alberhill, especially on the property of the Alberhill Coal & Clay Co., where the comparatively rough topography tends to favor the method by comparison with others. The method is applied under a wide range of conditions. Even in places where there is no overburden, topographic conditions may favor a glory hole, as a low-level tunnel can often be driven at small cost and mine cars can be cheaply loaded by gravity. Where the overburden is too thick to be economically stripped, an underground glory-hole system has found favor. Pillars are left at intervals to support the capping. On account of the complexity of geologic structure, and the irregularity of individual clay beds, the method can seldom be applied in a systematic manner at Alberhill, and costs vary widely from place to place.

UNDERGROUND MINING METHODS

Room-and-pillar Mining

Room-and-pillar mining, with round pillars, is the favorite underground method in use in California when underground mining becomes necessary. The size of the rooms and the thickness of the pillars depend on local conditions. In the Ione district, the method has been extensively applied to the mining of the Ione sand, which occurs in beds from 12 to 20 ft. thick, overlaid by variable thicknesses of volcanic breccia. At some properties, the thickness of the capping is such that surface methods are uneconomical. The sand is then mined by driving parallel tunnels at intervals of 25 to 30 ft. These are connected by crosscuts at the same spacing as the drifts. All openings are then enlarged both vertically and horizontally as much as possible without causing collapse of the roof. Rounded pillars from 8 to 15 ft. thick are thus left, at intervals of 25 to 30 ft. from center to center. In most instances in the Ione district, the openings are made large enough at the outset to accommodate auto trucks or wagons, which are loaded at the face by hand shoveling. In a few instances in this district, and elsewhere in the state, wheelbarrows or mine cars have been preferred.

The direct recovery of sand by this method, after robbing the pillars as much as is practicable, scarcely exceeds 60 per cent. of the total

available. There has been such an abundance of sand available in the Ione district that there has been little incentive up to the present to improve extraction, if it involves increased cost per ton of sand. In a few instances, the desire to decrease direct mining costs has led to excessive pillar robbing near the main entries, causing early collapse of the roof, which prevented access to sand at farther points until new haulage-ways could be driven.

The result of these mining policies over a number of years has led to the irrecoverable loss of considerable quantities of sand and has so seriously disturbed large areas of potential worth that the cost of opening these for future mining has been greatly increased. It is a pleasure to note that the necessity for conservation and systematic mining with the long-time view is being recognized in the district, and considerable improvement will undoubtedly be made in all future mining operations. On the other hand, it is unlikely that any real progress can be made unless higher prices can be obtained or larger outputs marketed.

The most obvious change of method that can be proposed for a property of long prospective life and steady production is the systematic layout of rooms in blocks of definite limits, to be mined as a unit. Development work should proceed to the far limits of the block before active mining is begun. Then the pillars can be cut back either by retreating from the boundaries, allowing the cut-back areas to cave naturally, or else by blasting. Some recovery of sand from the pillars can be expected, even after they are blasted down and the roof is caved. The use of underground shoveling machines is undoubtedly warranted at all properties producing more than 100 tons per day for four months of the year.

In lieu of the round-pillar method, it may be more satisfactory in some cases to use a drift or crosscut retreat method, as the direct loss of sand is usually less for the same degree of safety.

Other Underground Methods

Various modifications of room-and-pillar mining are in use, other than the round-pillar method already described. Of these, drift-retreating has found favor. In this method, as applied to flat-lying deposits up to 12 ft. thick, parallel drifts, usually about 5 by 7 ft. in section, are driven at intervals of 20 to 30 ft. to the working limit of the deposit. Loading is usually by hand into mine cars. Timber support may be necessary in places. The retreat is made by slabbing to the roof, and slicing a cut from each side of the drift pillar. An extraction of 75 per cent. or more can usually be obtained before the back becomes dangerous. The method is flexible in that the pillar support can be readily adjusted to conform to the strength of the back, and, if desired, a row of posts can be used alongside the tramming drift to support the back while

practically all of the clay is being mined from the pillars, the method then grading into an open-stope method, in which systematic timbering is used instead of pillars for support. This method is favored when the unit value of the clay justifies the extra mining cost entailed in obtaining practically complete extraction.

Where systematic timbering is used instead of pillars, the cheaper grades of round timber, from 6 to 12 in. dia., are generally satisfactory for temporary support. Posts are placed at regular intervals, usually 5 ft. center to center, and large headblocks are used. Framed tunnel sets, of first-quality timber, are often necessary in the haulageways that must be maintained through weak ground.

Where topographic conditions are favorable to a low-level tunnel entry, it is good practice to maintain a separate haulage level 10 to 20 ft. below the clay bed, to permit gravity loading of haulage vehicles through raises connecting with the floor of the clay bed. Mine operators will readily appreciate the factors that must be considered in determining the most economical spacing of the transfer raises.

SUMMARY AND CONCLUSIONS

1. The development of systematic methods for prospecting and mining clay in California has been hampered by various factors, chief among which are the following:

- a. Abundance of clay, leading to
 - (1) Low prices
 - (2) Severe competition.
 - (3) Lack of incentive to conservation of future supplies.
- b. Relatively small scale of many individual operations, which imposes a serious limitation on the money that may be spent for equipment.
- c. Irregular occurrence of clay, and a multiplicity of varieties, in some districts. This factor is in opposition to the abundance of clay, and is partly responsible for the relatively small scale of individual operations.
- d. Uncertain market, caused by
 - (1) Fluctuations in building activities from year to year.
 - (2) Risk of new discovery by a competitor of similar or better clays than those in use.
 - (3) Fickleness of architectural modes. A given product, such as roofing tile, may be immensely popular one year and little in demand the next year.
- e. Royalty-lease operation of many properties. Short-term leases discourage expenditure for equipment; long-term leases where clay is plentiful encourage haphazard mining.

2. More attention will be paid in the future to systematic prospecting and mining, designed to secure the highest net return from a given property, some of the controlling factors being:

- a. The larger manufacturers of clay products can now see a definite limit to the supply of cheaply obtainable clay from California sources. More properties are being acquired by purchase, leaving fewer to be operated under royalty-lease.
- b. Consolidation of manufacturing interests into larger groups has favored large-scale mining, has minimized competition, and has tended to stabilize the market; or at any rate has permitted a more accurate estimate of future requirements.

3. With respect to prospecting methods, shallow open cuts and test pits at irregular intervals are giving way to systematic drilling. Where the formations are soft and depths do not exceed 30 ft., hand augers are satisfactory. For more severe conditions, power-driven core drills are preferred.

4. Open-pit mining is generally preferable to underground mining if the thickness of the overburden is less than that of the clay; but where the clay has very low unit value, underground mining may be more economical if the ratio of stripping to clay exceeds 0.6.

5. In open-pit mines, hand shoveling is suitable for outputs of less than 30 tons per day. Scrapers are in use for outputs of 15 to 100 tons per day but are best adapted for outputs of 30 to 100 tons per day. Small revolving power shovels are preferred for outputs exceeding 100 tons per day, but if a new shovel must be bought for a property of short prospective life, a much higher daily output is necessary to justify the expenditure.

6. If underground mining is necessary, economic conditions seldom warrant the use of timber in lieu of pillars for support of excavations. From 25 to 40 per cent. of the clay is left in pillars and is generally unavailable for future mining.

7. Bins with drawing-off chutes are needed at a number of loading points, to replace simple platforms from which railroad cars are loaded. An efficient device is especially needed for loading box cars, this being now done almost exclusively by hand, at a cost of approximately 20 c. per ton.

DISCUSSION

W. McCLINTOCK, Los Angeles, Calif.—Professor Dietrich has given a very complete and interesting account of the manner in which prospecting and mining is done in the various clay-producing districts of California. There is not much that can be added by way of discussion except to bring out a little more in detail some of the points he has touched upon.

Clay deposits are usually discovered by observation of outcrops which, however, are apt to be more or less indistinct. The best time to look for these outcrops is

during the winter rains or immediately thereafter, as the rain washes away the loose soil which covers the outcrops and leaves the clay formation exposed. However, it is not always necessary to find outcrops in order to get an idea that clay beds may exist. The overlying formation may indicate that there is clay underneath as well.

For preliminary prospecting, the best method, as Professor Dietrich states, is to use a hand auger, although this cannot be used to depths over 20 to 30 ft. depending on the character of the ground. If no clay is found, we generally conclude that even if clay does exist it will be too deep to mine advantageously by open-pit methods and the prospecting is abandoned. If, however, for any reason it is desired to go further, a power drill must be used or a shaft may be sunk in what seems to be the most favorable point to test the formation.

In determining value of clays found in prospecting, drying and burning tests are the only satisfactory guide. The clay is crushed in the laboratory and made up into small test pieces, 4 to 6 in. long, enough water being added to permit proper molding. These pieces are dried and fired in a small furnace and the results observed. Burning must be done to certain specified temperatures which will approximate the temperatures to be employed in firing the wares for which the clays will be used. It is usually desirable to fire trial pieces to several different temperatures in order to get the range of temperatures within which it is possible to use the clay; that is, we should know the minimum temperature at which the clay is completely burned and the maximum temperature to which it can be subjected without fusing. These limits for a given clay constitute what is called its burning range.

In making these tests, drying shrinkage and burning shrinkage should be carefully noted as well as color, tendency to crack or warp and any other factors which affect the use of clay in practical operations. Chemical analyses, as a rule, are of little practical value, except in the case of fireclays where we want to know, in the first place, the alumina contents of the clay, and in the second place, the percentages of other impurities, especially iron and alkalis.

Professor Dietrich says that in the case of the sands of the Ione district, where underground mining methods are used, the pillars are permanently lost. In mining Southern California clays, this is not the case. Where underground mining is resorted to, the highest grade clays always are taken out and these are usually overlaid with other clays of lower grade. The latter may be mined later on by open-pit methods, and in such cases the pillars of the old underground mines are then reclaimed. As an example of this, in the early days the Douglass mine of Pacific Clay Products was mined as an underground mine with a recovery of around 50 per cent. of the clay. A few years ago, it was changed to an open-pit mine and two strata of overlying clays were utilized; also, all of the clay left in the pillars has been mined.

The cost of mining has been considerably more than would have been the case if the mine had been laid out and operated as an open-pit mine from the beginning.

One point might be noted—that in a number of cases, owners of ground containing possible clay deposits have retarded prospecting by asking exorbitant royalties. In the various eastern districts of the United States, where operations are on a royalty basis, the rates, as far as I am familiar with them, run from 5 to 15 c. per ton. In California, the rates paid have averaged considerably above this figure and, added to the high cost of transportation from the mines to the plants, if the latter are close to market, this has made the cost of clay in Southern California plants excessively high. If plants are located at the clay mines, the finished ware has to be transported to market, so that the cost enters in the same way. If owners of clay-bearing lands would take a more reasonable view of the royalty question, it would undoubtedly stimulate clay prospecting.

Mining and Preparation of St. Peter Sandstone in Arkansas*

By D. D. DUNKIN,† GUION, ARK.

(New York Meeting, February, 1928)

SANDSTONE has been prepared for glassmaking purposes, and marketed from the White River Valley in Arkansas at Guion, Izard County, since about 1910—soon after the completion of the White River Branch of the Missouri Pacific R. R. Co. Until 1921, operations were somewhat spasmodic and ended in many failures. In that year a plant was built by the Silica Products Co., Inc., and since that time the operation has been continuous and steady.

The land is held under leases, covering about 30 square miles of the best sandstone adjacent to the railroad. Most of the product is shipped to glassmakers.

GEOLOGY

The St. Peter sandstone is rather widely distributed in north central Arkansas and is exposed most prominently in outcrops along the slopes of the White River Valley and its tributaries that form the division between the Boston Mountains to the south and the Salem Plateau of the Ozarks to the north. It is of Ordovician age and is generally underlaid by Cotter dolomite and capped by Plattin limestone, except in the vicinity of Bellefonte and Everton. At those places, it is a massive cross-bedded, laminated, saccharoidal sandstone; underlying it is the Everton limestone and overlying it, the Joachim magnesium limestone.¹

The sandstone at Everton contains numerous phosphatic pebbles and inclusions of sandy limestone. The irregularity of cross-section, the evidence of sorting action and the phosphatic pebbles appear to be indications of secondary deposition.

The St. Peter sandstone occurs as a massive formation in the picturesque bluffs along the White River Valley from near the eastern escarpment of the uplift northwestward into the Ozarks. The exposure shows a series of gentle anticlinal and synclinal folds. The greatest thickness is at the southeastern end, where it is over 100-ft. in many places. It thins out to the north and west and then to isolated and scattered deposits which cap the ridges and mountains of the Ozarks.

* Outlining a rather radical departure from approved methods of drilling, placing holes, and blasting.

† President, Silica Products Co., Inc.

¹ U. S. Geol. Survey *Folio* 202: Harrison-Eureka Springs.

The sandstone is saccharoidal and is either white or cream-colored. The grains are small and well rounded and of nearly pure silica. This is also characteristic of the St. Peter sandstone found at Pacific and Klondike, Mo., and at Ottawa, Ill. The binder holding the grains together consists of clay and lime and is more frail than the grains. The glass sands of Pennsylvania, Maryland and West Virginia are in the Oriskany formation of the Ordovician age. The grains of the Oriskany are small and angular and the binder is siliceous. The Oriskany carries rutile, but the St. Peter has comparatively little.

The character of the St. Peter sandstone in Arkansas varies markedly even in short distances. Where it stands out in relief, without cover, the



FIG. 1.—RIPPLE MARKS. THE HAND IS PLACED TO SHOW THE GROWTH OF THE DEPOSIT.

presence of many impurities is indicated. The better sandstone is protected from rapid erosion only by the overlying limestone cap. Faulting, with considerable movement, is in marked evidence. The grain size, texture, and hardness of the rock is variable in any vertical section.

Where the deposit is suitable for commercial use it is massive and there are few cracks, fissures or seams. Water does not easily penetrate the deposit, though the sand is damp, as is evidenced by springs at the contact of the overlying limestone.

The St. Peter sandstone is generally accepted as a wind deposit, though this has been disputed. There are many markers at Guion that would indicate a water deposit. In working the deposit at Guion there is a vast difference between horizontal and vertical sections. The roof and floor of the mine follow very distinct lines of cleavage and the vertical sections show none. The roof and floor remain parallel. There are a few very distinct horizontal markers, such as color, grain size and bands.

There are very few ripple marks, though Fig. 1 would seem to show them. The hardness and cleavage vary in any vertical section. The boulders produced by blasting can be broken only by means of a wedge, and this



FIG. 2.—SHOWS HORIZONTAL MARKERS AND ABSENCE OF ANY VERTICAL MARKER.

along a horizontal plane of the boulder as it was *in situ*. Along this plane, even in thin strips, the rock will bend and show considerable deflection before breaking.



FIG. 3.—A PERSISTENT HORIZONTAL MARKER OF UNIFORM COARSE GRAIN. THERE MUST HAVE BEEN STRONG SORTING ACTION AT TIME OF DEPOSITION, AS IT IS COMPARATIVELY FREE FROM FINE GRAINS SMALLER THAN 65 MESH.

The results of chemical and physical tests on samples from the old quarry face at Guion, which is 80 ft. high, are shown in Table 1.

TABLE 1.—*Tests on Samples of Sandstone from Quarry Face at Guion*
Chemical Analyses

	PER CENT.		PER CENT.
Silica (SiO_2).....	98.90	to	99.61
Alumina (Al_2O_3).....	0.62	to	0.21
Ferric oxide (Fe_2O_3).....	0.04	to	0.02
Lime (CaO).....	0.42	to	0.08
Undetermined.....	0.02	to	0.08
	100.00		100.00

Sieve Analyses, on Tyler Standard Screens

	PER CENT.		PER CENT.
On 28 mesh (0.589 mm.).....	0.2	to	1.2
Through 28 and on 35 (0.417 mm.).....	9.47	to	21.0
Through 35 and on 65 (0.208 mm.).....	30.20	to	31.2
Through 65 and on 100 (0.147 mm.).....	31.52	to	10.0
Through 100.....	17.61	to	7.5

In glass-sand practice, fines are defined as grains that will pass through a 100-mesh screen. All above 5 per cent. are objectionable for glassmaking; the fines making seedy glass as well as being carried over by the draft to choke the checker work of the regenerators. The fines carry the bulk of the impurities.

CAUSES FOR CHANGE FROM QUARRYING TO MINING

Specifications for glass sand have become more strict on account of new machines for making window glass, and the placing of the manufacture under technical control. Keen competition has also had its effect. In addition to chemical purity, maximum and minimum grain sizes and grading from coarse to fine are important. "An ideal sand would be through 48 mesh (0.295 mm.) and on 58 mesh (0.223 mm.)." (Pebble.)

A good glass sand, in addition to being pure, will pass through 28 mesh (0.589 mm.) with not over 5 per cent. through 100 mesh (0.147 mm.) with *uniformity of grain size* from shipment to shipment. Still further, the specifications of some users call for 60 to 80 per cent. retained on 65 mesh (0.208 mm.) with graduation in grain size uniform from shipment to shipment.

These more strict requirements demand more careful preparation, and in order to continue with the quarry meant that the fines must be eliminated by some method. Fine screening was found to be a very difficult problem and also very inefficient. None of the reputable screen-makers cared to give any guarantees when screening was finer than 60 mesh. All tests showed that the elimination of 15 per cent. of fines meant the loss of 15 per cent. or more of coarser material. Without the handling charge, the loss alone would increase the cost about 50 per cent.

To obtain a sand to meet the specifications outlined, it was necessary to start a systematic campaign of sampling on vertical sections of all deposits available to the railroad. The best results were obtained from the samples taken in the old quarry. Here, down from the top of the deposit 12 ft., a seam 16 ft. thick was found, which was uniform and persistent. The average chemical and sieve analyses of the seam are given in Table 2.

TABLE 2.—*Analyses of Sandstone Seam in Old Quarry*

Chemical Analysis		PER CENT.
Silica (SiO_2).....		99.38
Alumina (Al_2O_3).....		0.57
Ferric oxide (Fe_2O_3).....		0.028
Lime (CaO).....		0.01
Undetermined.....		0.012
		<hr/> 100.00
Sieve Analysis		PER CENT.
On 28 mesh.....		1.2
Through 28 mesh and on 35.....		20.00
Through 35 and on 48.....		29.10
Through 48 and on 65.....		31.2
Through 65 and on 100.....		10.0
Through 100.....		8.5
		<hr/> 100.00

MINING METHODS

In order to get the sand from this seam, it was necessary to resort to mining. The entrance to the mine is by tunnel, the bottom of which is 20 ft. above the present quarry floor. The tunnel was placed to permit the grade to favor the loads and to provide natural drainage.

The general scheme leaves about 60 per cent. of the sandstone in the mine as pillars to support the roof. The tunnel, or drift, was driven straight in, 16 ft. high by 40 ft. wide. As in coal mining, rooms are turned off this tunnel every 100 ft., the turn being narrower than room width; corresponding to the room neck in coal mining. This neck is about 25 ft. wide at the beginning and gradually widens to 40 ft. The rooms are connected by crosscuts which are 35 ft. wide and are 80 ft. apart. The mine has natural ventilation and numerous exits. A convenient bend in the mountain permits the rooms or drifts to be driven to the outside.

There is a definite floor, to which all shots break. This floor is very hard and smooth, much like an artificial floor, though there are some undulations; it is composed of extremely fine sand grains, and particular care is used not to break into it. The roof follows a very definite line of cleavage; it is very smooth and much like the ceiling of a dwelling.

Drilling and Blasting

The drill used is the Waugh 93 jackhammer, with air by-pass; designated as "soft-ground hammer." The ordinary jackhammer, piston drill, or water Leyner drill is not adapted for this work because the binder is more frail than the grain, the grains are not pulverized and are of sufficient size to muck the steel. These types were all tried and discarded. Then a twist steel with an ordinary bull bit was used. It was better than the other drills but was not rapid enough. The rotation of the drill augured the cuttings from the hole.

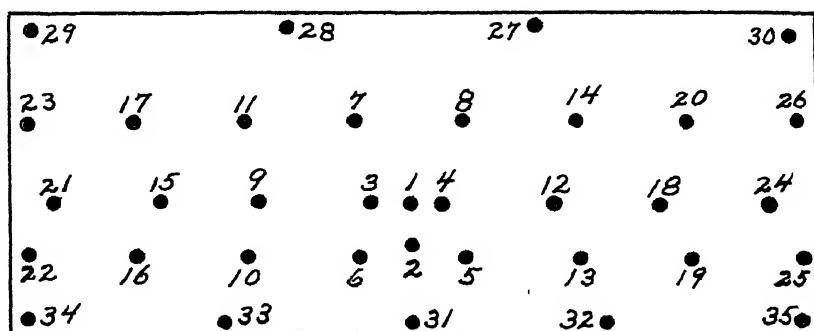


FIG. 4.—ARRANGEMENT OF HOLES AND ORDER OF FIRING. TOP AND BOTTOM HOLES DO NOT LOOK DOWN OR UP. SIDE OR WALL HOLES LOOK OUT OR GRIP 6 TO 8 INCHES.

The Denver Rock Drill Mfg. Co. assisted in working out this problem with twist drill steel and later brought out the Waugh 93 with air by-pass. With this drill it is essential to have a needle with a large hole and a large hole in the hollow steel. The steel in use is $\frac{7}{8}$ hexagon hollow steel with rose bit. The longest steel is 12 ft. and the gage is 1.5 in. The changes are every 2 ft., the gage enlarging by one-eighth.

The drill is operated by using the hammer for a few seconds, pressing a button, which stops the hammer, and by-passing the air around the hammer and down the hollow steel, blowing cuttings from the hole.

No breaks are maintained in room or drift. All holes are drilled straight into the face and all holes are 12 ft. deep. The holes next the pillar walls are gripped from 6 to 8 in. in 12 ft. Fig. 5 shows arrangement of holes and the order of firing. The top holes are drilled parallel with the roof and about 6 in. below it. The same method of drilling and placing is used for the bottom holes. None of the holes are squibbed to make a pocket for additional dynamite. The first few holes fired have from one to three sticks of 60 per cent. gelatin dynamite in the back of the hole. The remainder of the hole is loaded with 40 per cent. extra gelatin.

Holes 1 and 2 are fired first and make sufficient break to relieve Nos. 3, 4, 5 and 6. All the holes are fired with fuse and in rotation. No. 8 detonators are used. This round of holes yields from 400 to 500 tons of sand. If the steel is full length and the holes are fully 12 ft. deep, the tonnage is from 500 to 525 tons of sand. Little or no secondary or boulder popping is necessary.



FIG. 5.—FACE OF DRIFT READY FOR LOADING. NOTE ABSENCE OF BREAKS AND BUTTS FROM FORMER SHOTS; 12-FT. TAMPING POLE, AND HOLES NUMBERED IN ORDER OF FIRING.

The average for a year showed 0.77 lb. dynamite to the ton of sand. At the beginning of this year a heading was carried at the top of the drift, about 6 ft. high, and the rest was lifted as a stope. There was the extra expense of mucking out for the driller so that the dynamite used early in the year was 1.05 lb. to the ton of sand. By carefully measuring the



FIG. 6.—CLEANING UP DRIFT; LOADING INTO 2-TON BOTTOM-DUMP TIMKEN ROLLER-BEARING CARS.

drift, spotting all holes and using extra precautions to get all the holes parallel, the dynamite now used is 0.42 lb. per ton of sand.

This method of placing holes and shooting has the advantage of not shooting into pillars, because all shots are away from pillars. The roof is not "burned" and full of pot holes and the floor is very level. All pil-

lars have vertical walls and an absence of talus accumulations. In all, it is about as ideal a mine as could be imagined. After two years of experience, there is no weakening of the pillars and, if anything, they are much larger than necessary. It is not the intention to change them until more years of experience can be used as a guide.

The method of drilling is unique, as no post or mounting of any kind is used. The top of the adjustable horse is a 12-in. board with the broad side up. The drill is laid on the board, with the driller sitting behind and advancing the drill with his feet.

Usually two men operate the drill, though at times, after all the steel has been brought to the working face and pipe lines and connections made, each takes a drill to operate. One man has drilled 420 ft. of hole



FIG. 7.—DRILLER AT WORK; THIS IS ONLY MOUNTING USED FOR ANY HOLES. INTERMEDIATE HOLES ARE DRILLED FROM ADJUSTABLE HORSES.

on a shift, each hole being 12 ft. deep. The average drilling is about 240 ft. per shift of 8 hours.

This method of placing holes is a very radical departure from all approved methods, especially in getting the first break.

Order of Operations

The sandstone is shoveled into 2-ton, bottom-dump cars, two men to the car. All cars are equipped with Timken roller bearings. The track gage in the mine is 3 ft. The length of the car inside is 7 ft. and the width at the top is 41 in. The height of the car above the rail is 43 in. The distance between axles is 20 in. This makes a long low car and favors the shovelers. The grade in favor of the loads makes it necessary to equip all cars with brakes. The shovelers run their own cars to the storage bin outside the mine, leading a mule to the bin and using the mule to pull the empty back to the face. The storage bin is 100 ft. long and the

track extends its full length, thus permitting dumping anywhere in this distance.

The sand is removed from this storage by a hoe-type drag. The drag was designed by the company and is built at the plant. It operates below the track used for the dumping of the sand. The sand is dragged into a tram car and hoisted up an incline to the mill storage bin. The car automatically dumps into storage at the mill. Two men, one operating a Joplin-type drag hoist and the other a Joplin-type hoist, elevate 450 tons of sand per day.

From mill storage, the sand is fed to a 16-in. Blake-type crusher, thence to elevator and to 30-ft. American Process-type rotary dryer. The setting and design of the dryer has been altered from the original

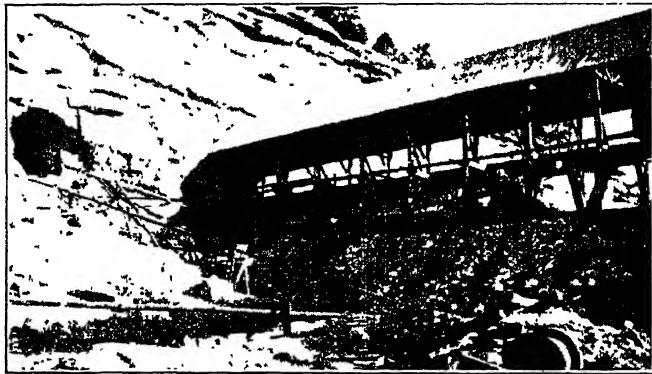


FIG. 8.—STORAGE BIN AT ENTRANCE TO MINE; NOTE SECOND OPENING AT LEFT.

design. The dryer is level instead of being on a pitch. To cause the sand to pass through, a series of wings are riveted to the shelves. These wings are in two sets placed 180° apart and shaped so that the sand falling or sliding on to them strikes an angle of 45°, causing the sand to be advanced. With this arrangement, the dryer at 9 r.p.m. handles 40 to 45 tons per hour. The dryer is oil-fired and complete combustion is obtained in the dutch-oven combustion chamber, so that only the products of combustion are used to dry the sand. From the dryer the sand is elevated and sent to two trommels in series, each 4 by 9 ft.

There is enough oversize in the sand to keep the screens from blinding, even though these screens deliver a product that is all through 28 mesh (Tyler standard screen scale). The oversize passes to a small set of Cornish rolls and is returned to the screens. The throughs are elevated and sent to the grader, which removes the fines so that a product can be guaranteed to contain less than 5 per cent. of them. The fines pass to waste and the finished sand goes to storage bins.

Railroad cars are carefully cleaned and paper-lined. Sand is loaded into them from storage bins by means of chutes, the storage being high enough to allow the sand to flow by gravity.

Steam is used as the prime mover. A common slide-valve engine drives the mill and a compound steam two-stage compressor supplies air for the drills.

THE SAND GRADER

The Silica Products Co., Inc., worked on the problem of fine screening for several years and had numerous tests made by the different manufacturers of screening equipment. The problem was the reverse of ordinary practice in that practically all the material passes over the screens with only 5 per cent. or less passing through. The wire cloth required was fine and frail, and even with the most intense vibration there was serious blinding. It was soon discovered that it was impractical to use cloth finer than 60 mesh, which did not give the desired separation. Also, the grains are too large to permit of successful air separation.

To overcome the difficulty, a machine was developed for grading fine dry sand. Much of the credit for the final design is due to Supt. W. O. Tilton. Patent has been applied for. During the search for a method of separation, experiments led to the belief that each size of grain had a different coefficient of friction and if the sand could be fed on to a suitable surface in a ribbon-like stream, the coarser grains would slide and roll and the finer would adhere to the surface. This reasoning developed the idea of moving the surface up an inclined plane and thus removing the fines.

The machine at present consists essentially of an endless belt operating over a system of rollers so that the slope and speed may be varied; also, different portions of the belt may have different angles of slope. The fines are discharged at the top of the belt and the coarse at the bottom, into suitable receptacles. The machine has a belt 7 ft. wide and a feed slot 5 ft. long. The capacity of each machine is 12.5 tons per hour. The fines will vary from 75 to 80 per cent. through 100 mesh. This makes the machine very efficient as compared to the ordinary method of fine screening.

Another design has been developed, and patent applied for, in which more than one belt or inclined plane is employed. The slope and speed of each is independent of the other and more than one separation is possible, as each belt takes the fines from the preceding one and makes a further division.

DISCUSSION

W. M. WEIGEL, St. Louis, Mo.—The operation discussed in this paper had many vicissitudes in its early days and many failures, partly due to the lack of markets in that territory and a good deal to mismanagement and inability to mine cheaply and compete with the more established producers of silica sand. The operations in the

St. Peter sandstone in northern Arkansas, which is about the southern end of that great bed, are much the same as those around the city of St. Louis and Crystal City and farther north. The properties and the characteristics of the sand are essentially the same at all these places. The southern sand is probably a little finer grained than that of the other districts, so it has not as yet been developed for other uses than silica sand, mainly glass sand; that is, no particular abrasive sands are sold, in the form of blast sand or filter sand. The latter could be done but the percentage of extraction would be rather low, so that no attempt has been made to use it except in the silica sand industry and to a certain extent to supply the sodium silicate markets.

Mr. Dunkin has not mentioned his markets. This operation supplies most of the sand for the glass plants in the South—Arkansas and Louisiana—and some of it goes to Western Arkansas, Oklahoma and Western Missouri; in fact, some shipments have been sent as far as Mexico. Why the buyers wanted that particular sand shipped so far, I do not know, but it may be for the same reason that glass plants in Los Angeles buy Belgian glass sand.

There is the happy advantage at this location of comparatively cheap labor which is quite efficient. The records of the drilling sometimes show an average of 250 or 300 ft. a shift by one driller, which is pretty good considering the conditions. Of course, the stone is not really hard but sometimes there is difficulty with the hole choking. They drill so fast that they cannot blow out the cuttings. A standard drill was tried, but the material was cut out so rapidly that it choked the hole, so a special hammer drill was developed with an unusually large needle and large hollow steel, which greatly improved the operations.

F. C. HOOPER, New York, N. Y.—On what basis is the labor paid?

W. M. WEIGEL.—I think it is day labor.

N. C. ROCKWOOD, Chicago, Ill.—Is the mine dry?

W. M. WEIGEL.—There is no water running through it, but the sand is damp. If you squeeze the broken sand in your hand, it will adhere, but there is no water running out of the mine.

F. C. HOOPER.—There is no dust in the drilling?

W. M. WEIGEL.—Not as far as I could see.

G. R. MANSFIELD, Washington, D. C.—Did they use more than the 16 feet?

W. M. WEIGEL.—No, that is all they used. The author has quarried more than that for steel molding but for glass sand he has taken only the 16-ft. bed beginning 12 or 14 ft. below the top of the St. Peter sandstone. The sand below that is sometimes a very fine grain—too fine to meet the specifications for glass sand.

H. J. BROWN, West Newton, Mass.—What was the tonnage?

W. M. WEIGEL.—He is putting out 300 tons a day, I think.

H. J. BROWN.—Is it an ideal shovel-loading proposition?

W. M. WEIGEL.—You mean for mechanical loading?

H. J. BROWN.—Yes.

W. M. WEIGEL.—They are loading sand underground with an electric shovel in some places, but not at this operation, which is comparatively small. With the cheap labor in the South, it is very doubtful even with a considerably larger tonnage, whether it would pay to use a mechanical shovel.

C. R. FORBES, Rolla, Mo.—What do they pay labor?

W. M. WEIGEL.—The muckers and shovelers probably get about 35 c. per hour. I would not be sure of that, but probably it is not any more. The drillers probably make about \$4 a day.

J. R. THOENEN, GREENVILLE, Ohio.—What kind of explosives do they use?

W. M. WEIGEL.—In the holes near the center, two or three sticks of 60 per cent. dynamite in the bottom of the holes.

F. P. KNIGHT, Beverly Farms, Mass.—On this class of material, not thoroughly dry, would the separation method work so well?

W. M. WEIGEL.—It is thoroughly dry. After it passes through the first crusher, it is run through a rotary dryer. Probably the moisture is not over 0.5 per cent.

Suggested Improvements for Smelting Copper in the Reverberatory Furnace*

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(New York Meeting, February, 1928)

THE development of the reverberatory furnace for smelting copper ores up to 1912 was described by E. P. Mathewson¹ with details concerning the great changes in dimensions of the furnace. Hayward² tabulated similar data, giving developments up to about 1924, in his revision of Hofman's Metallurgy of Copper.

DEVELOPMENT OF PRESENT SMELTING PRACTICE

Dimensions of Furnace

Very great changes were made in the dimensions of the smelting hearths of the furnaces in the period from about 1800 to 1906, the length increasing from about 11 to 116 ft., and the width from 8 to 19 ft. During this time the tonnage smelted increased from about 9 to 250 tons per 24 hr., and the tons of charge smelted per ton of coal increased from 0.77 to 4.3. Dr. Mathewson³ noted no great difference in the "fuel ratio" (*i. e.*, the amount of charge smelted per ton of coal) as the furnace at Anaconda was increased in length by stages from 85 to 116 ft. (The percentage of copper in the slag, however, was lowered from 0.42 to 0.36 per cent. The other components of the slag were not given.) Lump coal fired on grates was the fuel used during this period.

The practice in operating the furnaces over such a length of time naturally varied considerably, and there were differences in composition and nature of the charge, yet it may be considered that the improvements made were due largely to increases in the size of the furnace used. Doubtless some credit should be given to improvements in operating technique, like changes in the methods of firing the lump coal. However, the greater part of the advance was probably due to factors influencing the

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¹ E. P. Mathewson: The Development of the Reverberatory Furnace for Smelting Copper Ores. *Trans.* (1912) 44, 781.

² H. O. Hofman: Metallurgy of Copper. Revised by C. R. Hayward. McGraw-Hill Book Co., New York, 1924.

³ *Op. cit.*, 784.

percentage of heat lost; for example, the decrease in radiating surface per unit of volume of the heating chamber, with an increase in the size of the furnace.

If 116 ft. be taken as the preferred length of furnace in 1906, it can be said that for 20 years there has been little desire to change this dimension very greatly. Some large smelting companies think that a shorter reverberatory is better, and others have extended their furnaces to about 130 ft. in length. The objective in the latter case is to secure a better separation of the slag and matte rather than to increase the "fuel ratio."

One of the furnaces noted by Hayward⁴ as smelting 860 tons of charge per day, or 5.6 tons per ton of coal, has the approximate dimensions quoted for the 1906 furnace; *i. e.*, 19 by 116 ft. (For Garfield, Hayward gives 7.2 tons of charge as being smelted per ton of coal.) The change in furnace capacity since 1906 is not then due to changes in size of the hearth of the furnace, such as was noted over the preceding 100 years.

Other dimensional changes in the past 20 years have been an increase in the area of the uptakes, and to a lesser extent an increase in the height of the roof above the bath; in general, the provision for the handling of more gas within the furnace. These changes made it possible to increase the rate of burning coal.

Preparation of Charge

Preheating.—An increase in the intensity of firing, which permits a corresponding increase in the rate of charging, is the main reason for the more recent increase in the capacity of reverberatory furnaces. This increase in furnace capacity has been accompanied steadily by an increase in the amount of charge smelted per unit of fuel. The fuel now used, with few exceptions, is powdered coal or oil. The preheating of the solid charge—*i. e.*, feeding hotter calcine—has assisted greatly in increasing the amount the furnace can handle.

Concentration.—There are other factors not strictly related to smelting that must be considered. How small the particles of the charge may be crushed to advantage would seem to be a matter open to argument. It is admitted, however, that the finely divided charge employed at present smelts more easily than one containing crude ore and much jig concentrate, as with the finer charge more surface is exposed and the slag-forming constituents are very likely to be more intimately admixed. Other factors, such as the chemical composition of the charge, might be considered a constant for any particular ore, or dropped out of consideration as not being amenable to change by the smelter itself. Thus, the changes in the nature of the charge brought about by changes in methods of concentration, including such factors as lessening the tonnage to be

⁴ *Op. cit.* 196.

smelted by improving the grade of concentrate, might be disregarded from this consideration of methods to improve present reverberatory costs.

Methods of Charging.—As suits involving the use of charging methods are now under consideration by the courts, the methods of charging and the influence of changes in methods of charging upon smelting will not be considered. It is necessary to note only that larger capacities and better fuel ratios have been reported by users of both center and side-charged furnaces since the adoption of the system of more intense firing. It further appears that the side walls may be taken care of satisfactorily in either case as far as corrosion is concerned.⁵ We are then free to pass to the main purpose of this paper, which is to forecast along what lines it is most feasible to increase the capacity of the reverberatory, and the number of tons of charge smelted per unit of fuel.

POSSIBILITY OF FURTHER IMPROVEMENT

Furnace Dimensions

It is unlikely that the reverberatory will be lengthened. The increase in the width of the furnace was made chiefly to secure a larger hearth area, more capacity, and a roomier furnace to operate, with more space for the gases. (Later comparative tests with fine charges have shown that the chief improvement in operation is that the wider furnace with the larger bath of matte gives fewer periods of erratically high losses, rather than a lower percentage of copper in the slag at any time.) The roof is now in many cases supported entirely by the steel work, without resting on the walls; a few suspended arches are even in use. The factor limiting the widening of the furnace would seem to be only the compressive strength of the arch brick at high temperatures. The volume for the gas space and its possible enlargement, without sacrificing efficiency of utilization of heat, is receiving considerable study. Greater enlargement of the hearth area, however, is deemed inadvisable by many who believe that improvements in flotation methods may greatly reduce the tonnage to be smelted.

PREHEATING MATERIALS ENTERING FURNACE

Charge.—Among the methods of improvement left for consideration are those associated with an increase in the rate of burning fuel, and in the rate of adding fresh charge. With the latter is connected the preheating of the products fed to the furnace—charge, fuel, and air. In the reverberatory smelting, the *rate* of supplying heat to the charge is an extremely important consideration. As long as the charge is solid it is a fair insulator, whose layers heat through slowly, and will not “take” much

⁵ Frederick Laist: Quotation from Court Testimony. *Engng. & Min. Jnl.* (Oct. 16, 1926) **122**, 624.

heat; thus the rate of smelting is slowed down. It is obvious that the thicker the layers, the more important it is to preheat the charge, and much thought has been, and is being, given to various economical sources of heat for preheating the charge as nearly to the point of incipient fusion as is mechanically feasible.

Fuel and Air.—Practically nothing has been done on preheating the fuel, beyond heating the oil so that it will flow freely. Regenerative checker chambers were used on the gas-fired reverberatory furnace at Great Falls, Mont., but in general the adoption from ferrous metallurgy of the feature of preheating the air as in the steel open-hearth practice, or, in addition, of drying the air as in iron blast-furnace practice, has met with little serious encouragement. That is, preheating the air has been tried at several plants not requiring power⁶ from waste-heat boilers, but no general extension of the system is promised.

The attempt to lessen the fuel requirements for smelting by preheating and intimately mixing the charge, etc., is worthy of much work, as such methods would not require the use of other refractory materials than those employed at present. However, practically the whole charge to the reverberatory furnace, except the molten converter slag, is now put through the roasters; the calcine is carefully kept hot, and the charge in many cases is being spread out over the surface of the bath in thin layers, so that it heats quickly to the fusion point. Putting all the solid charge through the roasters mixes as well as preheats it, particularly when the siliceous flux is added in finely divided form like the rest of the material. The intimacy of admixture of the converter slag still leaves very much to be desired and a careful examination of the calcine may disclose surprising irregularities in its composition.

Rate of Firing

The increase in the rate of firing fuel and the establishment of a short, hot, smelting zone in the furnace are the factors that have been almost universally adopted in reverberatory practice, no matter how the other details of operation may differ. The adoption of this method of smelting has been the main cause of increasing three to four times the capacity of the reverberatory and of doubling its fuel ratio—i. e., tons of ore per ton of fuel—since the hearth dimensions of the furnace have been held

⁶ C. Offerhaus: Gas-fired Reverberatory Furnace at Sulitjelma, Norway. *Engng. & Min. Jnl.* (Dec. 25, 1915) 100, 1033.

R. R. Moore: Recent Reverberatory Smelting Practice. *Engng. & Min. Jnl.* (May 14, 1910) 89, 1021; (May 21, 1910) 89, 1063.

F. A. Leas: Regenerative Reverberatory Copper Furnace. *Engng. & Min. Jnl.* (Nov. 7, 1908) 86, 898.

W. G. Perkins: Non-reversing Regenerative Furnace for Copper Smelting. *Min. & Sci. Pr.* (Jan. 2, 1917) 114, 759.

approximately constant. The establishment of such a zone is, in itself, the main reason that hearth dimensions have not increased, since much of the furnace is now used chiefly as a settler. When a long flame was used, the temperature of which was not greatly above that required to melt the charge, the addition of fresh charge so lowered this small marginal range of temperature that the rate of smelting was lessened, even though the smelting was done over a long zone. When the short, hot, localized zone was established, although the temperature was increased not many hundred degrees, the available margin of temperature over what was required to fuse the charge was increased several fold, and the chilling effect of the fresh charge was greatly diminished.

The rate of transmitting heat to a charge depends on its receptivity to heat as well as on the temperature employed. To determine to what extent the rate of smelting is a function of temperature is a matter, then, that would require actual experimentation rather than calculations based upon thermochemical data. There is no doubt, however, that the transfer of heat by radiation between two bodies is proportional to the difference between the fourth power of the absolute temperature of the hot body, and that of the colder. Also, it is obvious that the amount of heat utilized in smelting is only the surplus left above the amount spent in satisfying heat losses, and requirements that are not very far from being constant per unit of charge, provided the charge is uniform in temperature, chemical composition, etc.

Present practice shows no indication, as yet, that the advantages to be gained from the continued increase in the rate of firing are soon to be halted by a lack of receptivity of the charge to heat.

The capacity may be increased either by extending the length of the present smelting zone, while maintaining the same temperature, or by increasing the temperature in the present zone. Experiments in present reverberatory furnaces have shown a great increase in capacity with increase in temperature, when the furnace was operated regardless of the life of the roof. Under present operating conditions—dusting, etc.—the practical point limiting firing is the temperature at which the arch brick drip rapidly.

REFRACTORY CORROSION

The corrosion of the brick in the arch is caused by the fluxing of the silica with the basic dust of the charge at high temperatures; "the brick do not burn out, they smelt out," as the smelter men say. The acuteness of the refractory problem at any smelter may be judged, then, by the extent or the degree to which these two variables co-exist. The operator particularly interested in refractories is he who must utilize the fuel in smelting the ore itself; *i. e.*, smelt the maximum amount of

ore per unit of fuel, as the use of large amounts of waste heat for power under his conditions is uneconomical. In this case the obtaining of a good fuel ratio is inseparable from putting through a large tonnage, so the rate of firing must be forced and a short hot smelting zone established. When, in addition to this condition, the operator must handle a finely divided, highly basic charge—like most flotation concentrate—that dusts badly, he will have the maximum amount of trouble with his refractories. The dusting action in the furnace may be greatly aggravated by high gas velocity caused by insufficient cross-sectional area or too much draft, or by poor methods of charging.

Control of Dusting

The lessening of the amount of dusting with a given charge would give a greater capacity to a furnace by increasing the length of a furnace campaign. If the dusting were decreased below a certain critical point it is possible that a somewhat higher temperature could be employed. Preheating should also decrease the dusting by diminishing the time the charge was in a nonliquid condition. Like preheating, the control of dusting would also allow betterments without requiring other than the silica brick now employed. Schemes like the Ambler⁷ side feeder, operating on the principle of an underfeed stoker, are familiar to nearly all smelter men. An apparatus was designed by Mr. McGregor whereby the fresh charge would be conducted by a pipe down to the older charge smelting within the furnace,⁸ very much after the system for conveying fine flue dust through a telescoping canvas bag down into a charge car. Other schemes, having in mind the physical nature of the calcine, like possibly a slight nodulizing, suggest themselves, and doubtless the subject of the prevention of dusting will continue to receive the attention it deserves.

It may be conceded at once that the development of a refractory material that will withstand a higher temperature in the presence of dust will meet with great discouragement, notwithstanding its importance and desirability. In the first place, under favorable circumstances some of the reverberatory furnace roofs have a very good life. "The reported life of a silica roof, in the hottest part of the furnace, namely, that portion extending from 5 to 35 or 40 ft. from the firing end, ranges from 3 to 6 months."⁹ Young¹⁰ reported that at the United Verde smelter, where

⁷ J. O. Ambler: New Method of Charging Reverberatory Furnaces. *Engng. & Min. Jnl.* (Jan. 29, 1921) 111, 226.

⁸ E. H. Robie: Pyrometallurgy. *Engng. & Min. Jnl.-Pr.* (Jan. 19, 1924) 117, 109.

⁹ Report of Committee C-8 on Refractories, A. S. T. M. Industrial Survey of Conditions Surrounding Refractory Service in the Copper Industry. *Proc. A. S. T. M.* (1926) 26, Pt. I, 256.

¹⁰ G. L. Young: The United Verde Smelter. *Engng. & Min. Jnl.-Pr.* (July 19, 1924) 118, 97.

the charge is comparatively coarse, 29 months had elapsed for one furnace without any repairs being necessary.

It can be seen that the technique of rapidly repairing the roof has been well worked out. The cost for refractories at present may also be reasonable under some circumstances. For the Humboldt smelter, Corwin¹¹ gave costs for refractories over more than a year's operation that seem very reasonable—from 4.4 to 8.23 cents per ton of charge smelted.

Other smelters possibly have less favorable conditions, but the person recommending other refractories is very justly told by the smelter men that "Silica brick are good, and they are cheap." Nevertheless, the silica brick are being operated at the highest temperature they can withstand under the present conditions of operation, and it would appear that the properties of a few brick in the arch in the "hot zone" of the furnace are limiting the further improvement of reverberatory practice by making it impossible to increase the temperature.

Operating Details

No very systematically taken data on temperatures within the furnace have appeared in print. Textbooks quote 1650° C. as the temperature required for "slagging in" a new silica bottom for the furnace, and Hayward¹² gives gas (flame) temperatures from 1410° to 1650° C. in the hot zone of the furnace, the variations being probably due to the objects observed with the pyrometers employed—parts of the flame, incandescent brickwork, etc.

Before considering the refractories themselves, a few notes might be given on features of operation, other than the method of charging, which might affect the life of the brick.

The air pressure on oil and coal may vary from about 4 oz. to about 13 lb. The air pressure for the oil is usually between 6 and 12 lb. There is no marked conformity of opinion as to the optimum air pressure as far as the life of the arch is concerned. In general, it is believed that higher pressures will cause the more severe corrosion. The number of burners also varies greatly—from 4 to 13 have been quoted for oil. They are used in one horizontal plane, or in two superimposed planes, and are largely some modification of the Steptoe burner. It is commonly agreed that the flame should not impinge upon the charge or the roof. In the ideal case, probably the burners should all be parallel; but in smelting a fine charge some operators point the end burners inward and the other burners slightly downward. It is usually believed that coal gives a longer and "softer" flame than oil. However, the scouring action of the coal dust is thought to be severe if it is forced into the furnace at

¹¹ F. R. Corwin: Reverberatory Furnace Repairs. *Engng. & Min. Jnl.-Pr.* (May 31, 1924) 117, 889.

¹² *Op. cit.*

too great pressure. For this reason a pressure of between 13 and 17 oz. is often preferred—just about enough to get the coal properly into the furnace.

The original oil burners were little more than straight pipes. Then the design was changed so that the oil was atomized to a considerable extent; the atomization was carried so far with some burners that the brick in the hot zone were corroded too rapidly. With the present type of burner, the atomization of the oil is carried to the point where the hot zone is maintained at the desired length, and at a temperature that the roof brick can withstand for a reasonable length of time. For coal, the difficulty is usually to get the material sufficiently fine. Any trouble with the grinding mechanisms that results in feeding much coarse coal to the burners will cause a loss of the localized "hot zone," and the formation of a longer zone at lower temperatures, much after the type obtaining when lump coal was fired on grates. Examination with the microscope has shown that much of the pulverized coal used at present is about 700 mesh and from 85 to 90 per cent. is minus 200 mesh. Improvements in grinding the coal are being made steadily.

Survey of Service Conditions

The conditions met by refractories in the hot zone—dusting, temperature, etc.—and the properties of the refractories themselves are the factors limiting the intensity of firing. As this is one of the most promising of the few feasible methods for improving reverberatory practice, a preliminary survey was made of the field. A few samples of brick, were collected for study, operating conditions were observed, and it was intended to parallel later the study made for the steel industry.¹³

From a study of the behavior of the brick under operating conditions, manufacturers of refractories and operators may obtain a good idea of the conditions to be met in securing a better brick. A few observations from articles already published are used to supplement this study, without any reference to the merits of various styles of charging or to the operating data associated with them. A wealth of data on these latter points will probably be available in the near future. Therefore the building up of magnetite bottoms in the furnace will be omitted, even though the length of furnace campaigns at times may be almost as much limited by this factor as by the properties of the refractories. The corrosion of the roof brick only will be considered.

Details of operating practice, such as noted above, exert a great influence upon the life of the silica brick in the roof. A few points in construction that affect refractory service will also be considered before

¹³ B. M. Larsen, F. W. Schroeder, E. N. Bauer and J. W. Campbell: Service Conditions of Refractories for Open-hearth Steel Furnaces. *Min. and Met. Investigations*. Carnegie Institute of Technology, *Bull.* 23 (1925).

passing to the chemical factors like slagging and dusting and their variations within the furnace.

Besides differences in furnace dimensions, the construction of various furnaces differ in the quality of the brick used, the workmanship of the masons laying the brick, strength of the reinforcing steel, and whether or not the roofs are ribbed.¹⁴ A partial bibliography on the properties of silica brick is given in articles by Larsen, Bauer, Schroeder and Campbell. To the masons constructing the furnaces, the point of greatest importance is to have brick with straight edges and of uniform size.

Construction

The main problem of construction is to secure a furnace that is as tight against the entrance of air as possible, and yet provide for expansion upon heating. For instance, to prevent the entrance of air as much as possible, a mixture of sodium silicate and silica, or silica and clay, is swept over the top of the roof so that it may lodge in the cracks between the brick. From the work of Grieg¹⁵ it would be judged that for gouting or luting, finely ground silica bonded with a *little* lime would be preferable, and that the use of clay and alkali in particular should be avoided. To help prevent opening up cracks between the brick by the warping of the roof after the furnace is heated, the steel-plate supporting skewback is sometimes made continuous; *i. e.*, it is not riveted to the vertical I-beams that take, together with their tie rods, the thrust of the arch. The plants having charges that make for long life in the furnace arch usually increase that life by solidity of furnace construction.

The reverberatory copper-smelting furnace is seldom rebuilt as a unit. The corrosion of the refractory material is very unequal in the length of the furnace, decreasing rapidly with the distance from the bridge wall. The section of the roof over the hot zone of the furnace is replaced at least seven times more often than that over the slag bay. Therefore the roof is usually built in sections, any one of which can be replaced without disturbing the remainder.

Appearance of Corroded Roof

The inside contour of the roof of a reverberatory furnace that has been in operation for some time is very irregular. Each of the six to eight sections may have been replaced at different times and exhibit

¹⁴ O. E. Jager: Prolonging the Life of the Roofs of Reverberatory Furnaces at Anaconda. *Min. & Sci. Pr.* (July 19, 1919) 119, 85.

¹⁵ U. W. Grieg: Immiscibility in Silicate Melts. *Am. Jnl. of Sci.* (Jan. & Feb., 1927) 13, 1-44, 133-154.

great differences in the amount of corrosion. Frequently offsets of 6 to 10 in. occur between adjoining sections. It is difficult to make the section joints sufficiently tight to exclude the air. The worst part of the corrosion is always found round the places where the air has been allowed to enter—around the opening for converter slag, between the sections of the arch, between loosely placed brick, or poorly shaped brick with crooked edges.

Wherever oxygen and charge get together in the presence of silica brick these latter are corroded. In the presence of silica and excess sulfides, the iron sulfide probably oxidizes as far as FeO , in which form it attacks the silica brick and slags as $x\text{FeO} \cdot y\text{SiO}_2$; the composition is a function of the temperature, as well as of the analyses of the interacting solids. (It will be noted later that other bases, like copper oxides, are also present in the slagged brick.) At the higher temperatures found in the firing zone, a lesser amount of bases (FeO , Cu_2O , CaO), etc., is required to flux the silica brick sufficiently to cause them to drip. The slagging action of dust on silica brick is greatly accelerated by an increase in temperature, even more than would be expected from the increase in the speed of reaction.

There is a marked difference in the appearance of the brick from the various sections of the roof. The bricks over the slag bag are covered more or less uniformly with a black or blue-black glaze, from which small drips hang. This section of the furnace roof is similar in appearance to the roof of an open-hearth steel furnace under "normal operating conditions."

The corrosion of the refractories in the firing zone is much more equal. "Stringers" of a light brown color from 1 to 6 in. in length are frequently seen hanging from the roof. These "stringers" are evidently viscous at operating temperatures, indicating that an actual fusing of the fluxed refractory is taking place rather than the comparatively slow surface corrosion of the slag bay. The firing zone has the appearance of a badly overheated section of an open-hearth steel furnace roof.

Slag Penetration

Upon examination of the individual bricks we find that the depth of slag penetration decreases with the distance from the bridge wall, or what is more to the point, with decreasing temperature.

The length of the bricks themselves affects the depth of slag penetration to a considerable extent, due to differences in temperature gradients. The bricks shown in Fig. 1 were taken from the same section of a furnace, about 50 ft. from the bridge wall, and illustrate the effect of temperature gradient upon slag penetration. The shorter the brick, the sharper is the temperature gradient, and the less the slag penetration.

TABLE 1.—*Depth of Slag Penetration in Various Roof Sections*

Sample Number	Length of Brick, Inches	Distance from Bridge Wall, Feet	Days of Service	Depth of Slag Penetration, Inches
1	12	5	173	5.5
2	9.5	15	132	4.5
3	6	30	132	3.0
4	10	50	132+	4.0
5	10.75	110	325	1.75
6	9	Flue		1.5
7	19.5	5		7
8	16.5	10		6
9	4.5	15		2
10	7	20		1.5
11	9	30		3
12	8	35		3.5
13	10.5	40		3
Verb.	9	115		4

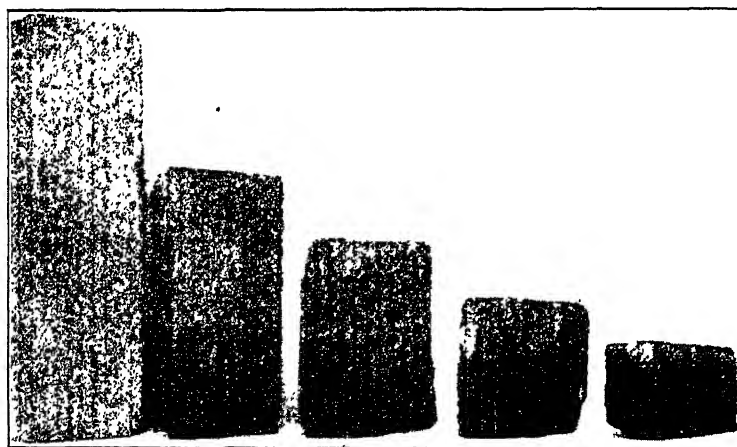


FIG. 1.—EFFECT OF TEMPERATURE GRADIENT ON DEPTH OF SLAG PENETRATION.

Composition of Brick-fluxing Materials

In considering the chemical attack of slag upon the brick, it is customary to show analyses of the slags present within the furnace. These analyses are of more interest to those studying the corrosion along the side wall as there is rarely enough splashing for the slags to come into contact with the roof. However, Tables 2 and 3 will give an idea of the ranges of the slags present in the furnace.

TABLE 2.—*Typical Analyses of Reverberatory Slags*

Substance	Approximate Range, Per Cent.	Usual Range, Per Cent.	Remarks
FeO	25-50	30-45	All Fe present calculated to FeO
CaO	2-24	2-6	Rarely over 10 per cent.
Al ₂ O ₃	5-12	8-12	Na ₂ O, K ₂ O, MgO, as compounds; ZnO also present
SiO ₂	32-50	35-44	PbO, etc., present in minor amounts

TABLE 3.—*Typical Analyses of Converter Slags*

Substance	Average Analysis, Per Cent.	Usual Range, Per Cent.	Remarks
Fe	48.0	40-56	Fe present as several compounds
CaO	0.3		
Al ₂ O ₃	3.0		
SiO ₂	22.0	18-30	

In the absence of samples of dust actually taken from the furnace while in operation, perhaps the next best idea of what the corroding dust is composed of may be had from analyses of the corroded slag brick. Analyses of the charge as given in Tables 4 and 5 are also of value, particularly those of the flotation concentrate. This product, after roasting, becomes the most finely divided constituent and would, therefore, seem likely to do the most dusting.

TABLE 4.—*Analyses of Reverberatory Furnace Charges*

Substance, Per Cent.	Hayden ^a	El Paso ^a	Tacoma ^a	Copper Queen ^a	Inter-national ^a	Nevada Cons. ^a	Ana-conda ^a	United Verde ^b
Cu	16.0	17.1	18.2	8.5	23.5	11.4	9.8	5.8
Fe	31.5	25.0	21.7	29.5	22.3	28.0	26.8	36.5
CaO	3.9	4.0	3.0	1.6	3.6	5.6	4.2	1.4
Zn								2.0
Al ₂ O ₃	6.0	5.3		7.0	5.1	7.2	5.6	7.7
SiO ₂	22.0	27.3	12.8	27.0	14.6	31.9	24.5	27.0

^a H. O. Hofman: *Metallurgy of Copper*, 193. Revised by C. R. Hayward. McGraw-Hill Book Co., New York, 1924.

^b G. L. Young: The United Verde Smelter. *Engng. & Min. Jnl.-Pr.* (July 19, 1924) 118, 97. This contains no concentrate; it is all crude ore.

TABLE 5.—*Analyses of Flotation Concentrate**

Substance, Per Cent.	Ray Cons. ^a	Copper Queen ^b	Nevada Cons. ^c	Cananea ^d	Nacozari ^e
Cu.....	19.8	12.75	14.9	19.0	28.0
Fe.....	19.6	27.1	24.6	29.0	29.4
SiO ₂		15.6	20.0	5.7	4.8
Insoluble.....	28.9				
Al ₂ O ₃		5.1	5.1	2.0	2.0
S.....		34.6	29.0	34.9	33.7
Through 200 mesh.....	83.0	Nearly all	65.0	57.4	71.2

* These analyses of flotation concentrate are not taken for the same year, and since flotation practice is changing very rapidly, they cannot be strictly compared. The copper from the first three mills was largely in the form of chalcocite, and from the last two in the form of chalcopyrite, which will explain in part some of the differences. For instance, the "insoluble" material will usually be higher with the more altered rocks containing chalcocite.

^a W. I. Garms: Remodeled Flow Sheet at Hayden Plant. *Ariz. Min. Jnl.* (Mar. 15, 1926) 9, 5.

^b E. Wittenau: The New Copper Queen Concentrator. *Engng. Min. Jnl.-Pr.* (May 24, 1924) 117, 836.

^c A. B. Parsons: Nevada Consolidated Copper Co. IV. *Min. & Sci. Pr.* (Sept. 3, 1921) 123, 323.

^d A. T. Tye: Differential Flotation of Copper at Cananea. *Engng. & Min. Jnl.-Pr.* (Apr. 10, 1926) 121, 597.

^e W. T. MacDonald: Selective Flotation at Nacozari. *Engng. & Min. Jnl.-Pr.* (Sept. 20, 1924) 118, 445.

In regard to the kind of flotation concentrate that may be expected in the future, it may be said that the present trend is toward even finer grinding, in the endeavor to free the finely disseminated copper minerals, and to make the composition of concentrate approximate as closely as possible that of the copper minerals of which it is composed. Just now the attempt is being made to "drop" more of the iron minerals, which are overbalancing the "acidic" minerals from the standpoint of making a good slag. An estimate of the extent to which a particular concentrate may be enriched by "dropping" the iron may be made from an examination of the chemical and physical associates of this substance. It would seem that the corrosive properties of the dust formed in many reverberatories will not be lessened for some time, for even if the iron content of the concentrate is lessened, it is almost certain that the dust will be finer in size, and copper oxide, as will be seen later, is a very corrosive base.

Dusting.—The subject of dusting within the furnace itself, and of the composition of the dust near the roof in the hot zone, has seemingly received very little study. With few exceptions, the feed to the large copper reverberatory furnaces is practically flotation concentrate, to which has been added siliceous material of lower grade in copper for

fluxing "excess iron." Although the total charge is usually more or less mixed and preheated in the roasters, as described, in most plants the siliceous material added as flux is much coarser than the rest of the charge. It would be thought that the kaolin and sericite of the original flotation concentrate, with the more finely divided sulfides (these latter, of course, altered by roasting) would form most of the dust. From the proportions of the various fine dust-forming constituents present, it would also be judged that the dust would be highly basic. For the copper minerals, the fineness of grinding required to liberate them would, of course, determine their sieve size—if chalcocite and chalcopyrite were present in the same ore, it would be thought that the chalcocite, being the more friable, would be the more finely divided.

Slag Drips.—It was found that most of these suppositions were checked when samples of slag "drips," taken along the whole length of the furnace, were analyzed. Also, composite samples of the brick were analyzed and the bases absorbed from the furnace dusts computed. The evidence thus obtained is confirmatory of the selective nature of dusting. The samples of drips from the roof taken at intervals from the firing end to the slag bay had the compositions and fusion points shown in Table 6.

TABLE 6.—*Analyses of Drips*

Sample Number	Location	SiO ₂ , Per Cent.	Fe ₂ O ₃ , Per Cent.	FeO, Per Cent.	Al ₂ O ₃ , Per Cent.	CaO, Per Cent.	Cu Calculated to Cu ₂ O, Per Cent.	Cone Fusion Temperature	Approximate Degrees F.
1	Plant No. 1, ^a firing end.	70.7	0.8	7.7	5.9	1.0	8.3	13-14	2600
2	Plant No. 1, 50-60 ft. from firing end.....	58.7	5.8	9.4	6.6	2.6	9.3	9	2390
3	Plant No. 1, 75-80 ft. from firing end.....	68.6	0.6	8.5	5.3	1.4	10.6	9	2390
4	Plant No. 1, uptake....	60.9	1.1	7.3	7.6	1.5	12.5	8-9	2370
5	Plant No. 2, ^b 45-50 ft. from firing end.....	60.6	2.9	4.5	16.9	3.6	1.0	9-10	2410

^a Charge mostly finely ground flotation concentrate from chalcocite ore.

^b Charge comparatively coarsely crushed chalcopyrite; siliceous material, highly micaceous.

It would be necessary to obtain a great many samples of drips from many furnaces in order to have the composition of the various samples show characteristics with any uniformity. However, each of the samples listed is a composite of 15 to 50 drips. Overlooking some irregularities, it may be seen that the copper in the drips increased as the slag bay was approached, the alumina and lime increased slightly, and the ferrous iron stayed about constant.

The fall in temperature in traversing the furnace from the firing end to the slag bay, of course, caused an increase in amount of bases necessary to react with the silica brick to form a slag of sufficient fluidity to drip

from the roof, but the varying portions of the bases present showed a preferential action in dusting. The finely divided micaceous flakes of sericite probably would be carried easily in the gas stream, and would, therefore, "dust preferentially." Analyses for alumina would offer a means for following the course of the sericite. The uncertain volatility of potash under the conditions obtaining when the drips were formed



FIG. 2.—ZONED BRICK.

would make it hard to use analyses for this base for a similar purpose. Sample 5 shows that when the sulfides of iron and copper are coarser, and the siliceous material is finer, the slag formed on the roof is of a very different nature. Plant No. 2, from which Sample 5 was taken, is noted for the long life of its furnace roofs.

When examined under the binocular microscope, the "drips" showed unfused particles of dust adhering to a glassy surface where fusion had already taken place. The dust particles probably attach themselves to the softened roof as flies adhere to sticky paper.

Samples 2, 3, 4 and 5 were all composed of drips having much the shape of drops of water held to the underside of a flat horizontal surface by surface tension. They were glassy-black in color and had a conchoidal fracture. It is evident that the roof surface over the slag bay was covered with a thin black glaze, very fluid at operating temperatures, which slowly dripped into the bath, but which was being constantly replaced by the fluxing action of the fume and dust on the silica brick. A certain amount of this slag was drawn up into the porous silica brick by capillary

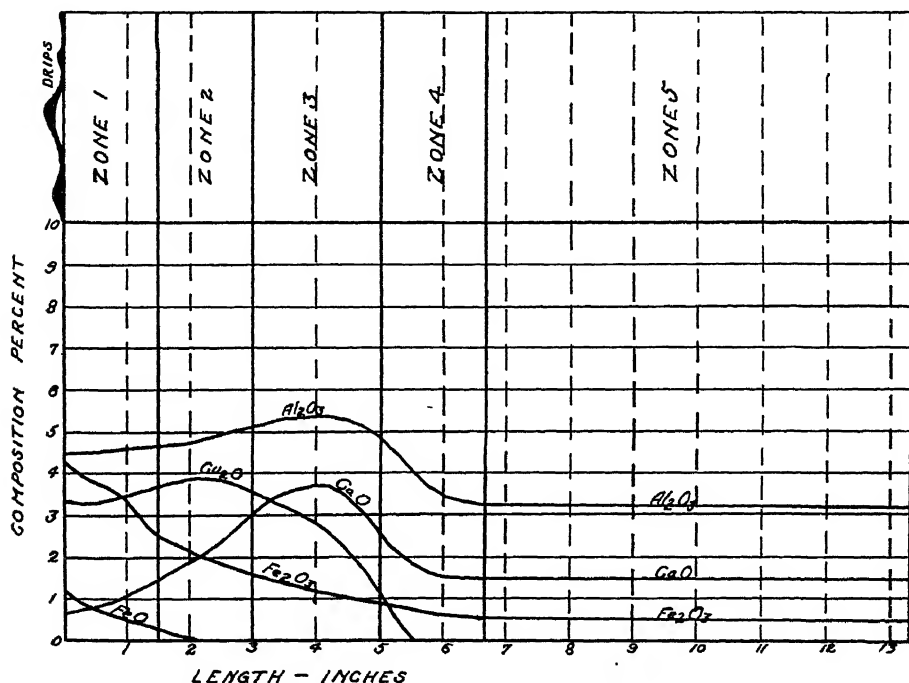


FIG. 3.—COMPOSITION OF ZONES IN USED BRICK FROM REVERBERATORY COPPER-SMELTING FURNACES.

attraction, the limit of penetration marking the point in the temperature gradient at which the slag was no longer fluid.

The drips composing Sample 1 (firing zone) were several inches in length and when broken showed a structure very similar to that of the first zone of the brick from which they were formed. The brick in the firing zone were fused together and the appearance of the roof in this section of the furnace indicated a fluxing and fusing of the refractory material rather than a slow surface corrosion.

Zoning.—A brick taken from the center of the arch about 50 ft. from the firing end of a furnace was found to have a fairly well developed zone structure. It was divided into five zones, as shown in Fig. 2, and each

zone was sampled for chemical analysis. The results of these analyses are given in Table 7, and are shown graphically in Fig. 3.

TABLE 7.—*Analyses of Brick Zones*

Samples	SiO ₂ , Per Cent.	Fe ₂ O ₃ , Per Cent.	FeO, Per Cent.	Al ₂ O ₃ , Per Cent.	CaO	Cu Cal- culated to Cu ₂ O, Per Cent.	Cone Fusion Temper- ature	Approx- imate Degrees F.
Drips.....	60.8	3.0	9.1	7.3	1.6	8.7	11	2462
Zone 1.....	83.8	3.8	0.7	4.6	0.8	3.4	26	2912
Zone 2.....	86.2	2.1	Trace	4.8	2.1	3.9	28	2975
Zone 3.....	86.7	1.2	Trace	5.3	3.7	2.8	28	2975
Zone 4.....	93.6	0.7		3.5	1.5		30-31	3050
Zone 5.....	93.6	0.5		3.2	1.5		31-32	3090

As in the case of roof brick from open-hearth steel furnaces¹⁶ there seems to be a concentration of lime and alumina in Zone 3. Lime forms

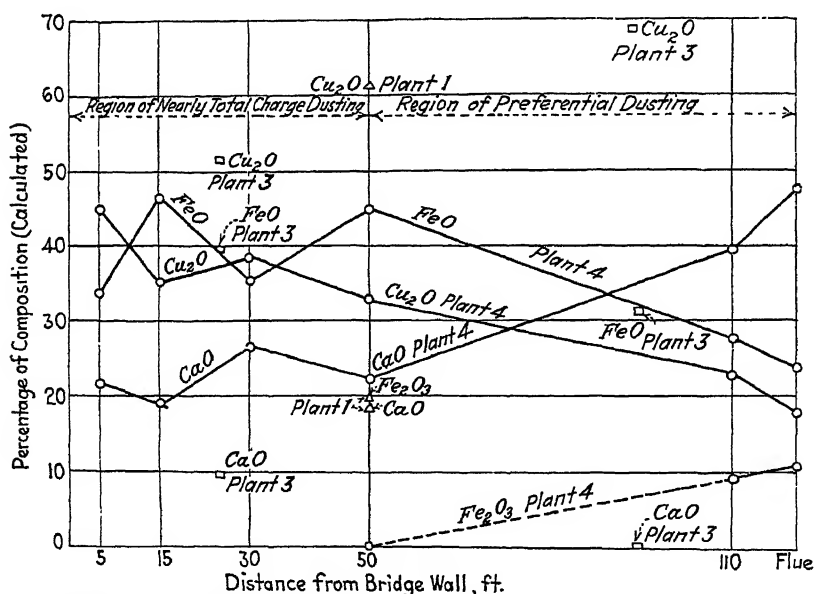


FIG. 4.—RATIO OF BASES ABSORBED BY ROOF BRICK IN VARIOUS SECTIONS OF FURNACE.

very fluid slags with flue dust and silica, and these slags may be drawn easily up into the brick by capillarity, absorbing more lime and becoming increasingly more fluid as they penetrate. If this occurs, the slags finally would form an enriched layer when their freezing temperatures were met. The zones left behind would contain undissolved silica and lime-free slag. However, copper is present in the first three zones in decreasing amounts and causes the fusing points of Zones 1 and 2 to be lower than in the case

¹⁶ Larsen, Schroeder, Bauer and Campbell: *Op. cit.*

of the open-hearth brick. Zones 4 and 5 are of nearly the same chemical composition, indicating that Zone 3 is the limit of slag penetration, although Zone 4 differs in color from Zone 5.

This penetration of the slag into the brick seemingly caused the gradual solution of the quartzitic grains, as is indicated by their rounding in Zones 3 and 4, and their complete disappearance in Zones 1 and 2. A plane of weakness is found at the junction of Zones 3 and 4, or rather at the limit of slag penetration. Differences in the mineralogic structure with the corresponding difference in expansion is probably responsible for this plane of weakness. In furnaces that have been cooled for repairs many brick are found which have spalled or broken at or near this plane.

The approximate ratio of the bases Fe_2O_3 , FeO , CaO and Cu_2O in the slags absorbed by the brick in different sections of the furnace was calculated from the chemical analyses of composite samples of unused and slag-saturated brick, together with their weights and volumes. These approximate compositions neglect silica and alumina, because they were used as the basis for calculation. The calculated ratios of the bases absorbed by the brick are given in Table 8 and are shown graphically in Fig. 4. No sulfur was found in the brick.

TABLE 8.—*Calculated Ratios of Bases Absorbed by the Brick*

Sample from:	Distance from Bridge Wall	Fe_2O_3 , Per Cent.	FeO , Per Cent.	CaO , Per Cent.	Cu Calculated to Cu_2O , Per Cent.
Plant No. 3.....	Firing end		38.7	9.8	51.5
Plant No. 3.....	Slag bay		31.2		68.8
Plant No. 4.....	5 feet		33.9	21.8	44.3
Plant No. 4.....	15 feet		46.1	18.8	35.1
Plant No. 4.....	30 feet		35.4	26.3	38.3
Plant No. 4.....	50 feet		45.0	22.3	32.7
Plant No. 4.....	110 feet	9.2	27.8	39.8	23.2
Plant No. 4.....	Flue	10.3	24.0	47.7	18.0

TABLE 9.—*Calculated Ratios of the Bases in the Calcines*

Plant	Fe Calculated to Fe_2O_3 , Per Cent.	CaO , Per Cent.	Cu_2O , Per Cent.	Copper Minerals Approximate Per Cent.
Plant No. 1.....	67.2	5.8	27.0	90 % chalcocite 10 % oxidized
Plant No. 2.....	87.0	2.3	10.7	10 % chalcocite 90 % chalcopyrite
Plant No. 3.....	51.6	5.8	42.6	90 % chalcocite 10 % oxidized
Plant No. 4....	56.3	20.1	23.6	50 % chalcocite 50 % chalcopyrite

It is at once evident that the composition of the fluxes absorbed by the refractories is not the same throughout the length of the furnace, and therefore must be caused by the preferential dusting; certain constituents are carried farther by the gas stream, due to their physical properties, such as size, shape and specific gravity. The ratios of the above constituents in the calcines of these plants are given in Table 9.

Apparently the finer constituents of the charge dust preferentially, and the corrosion of the brick is in part a function of the size and composition of the materials of the charge, particularly of the relative sieve size of the acidic and basic constituents.

CONCLUSIONS

Preferential dusting occurs to considerable extent even in the hot zone of the furnace. Of the bases, the percentage of the lime and the copper in the brick is larger than would be assumed from the analysis of the charge, and the percentage of iron is smaller. The probable mechanism of the fixing of FeO as $x\text{FeO}\cdot y\text{SiO}_2$ has been described; it is also probable that the copper acts somewhat similarly. The fluxing action in general may be summarized as follows: The pore space of the brick draws up these fusible silicates by capillarity, and the slag enters to a distance determined by its fluidity, which is in turn governed by the temperature. Where the fluxing is not too rapid zones appear in the brick. On the bringing up of fresh basic dust, the silica content, particularly of the finer sized binder, becomes sufficiently lowered for the bottom part of the brick to become fusible and droplets to appear in the hot zone where the action is very rapid. These drips contain noticeable fragments of silica altered only by heat treatment.

The maximum practical temperature of operation of the furnace is determined by the point at which the silicates made by the dust in contact with the roof become easily fusible. The temperature where the drips from the hot zone of one furnace fused was 2600°F . (about 1430°C). It was also noted that coarse uncombined particles of silica will even slough off. The lessening of dusting will decrease the speed of corrosion of the brick at a given temperature. The extent to which the dusting must be reduced before an attempt may be made to increase the temperature and thereby the capacity markedly, is another very important point upon which data are lacking, and results may be conjectured only. Unless the dusting were nearly eliminated, there would be the same phenomena of slag penetration of the brick and lowering of its fusion point. As the temperature is increased, the base "requires more silica to satisfy itself" and after the binder goes, large unfused lumps of silica may form the drips. The length of increase in life of refractories on a diminution of the amount of dusting at the present smelting temperature cannot be

used, then, as a basis for computing the increased length of life of brick under conditions when the dusting is likewise decreased but the temperature is raised.

Judging from the experimental results of steel open-hearth practice, cooling the silica brick with water jackets would not greatly lengthen their life. The effectiveness of water-cooling with refractories of higher heat conductivity (magnesite, etc.) would probably depend on the intimacy of contact between the water jacket and the refractory.

From the historical development of the reverberatory it would seem that the best chance of improving present practice lies along the line of still higher temperatures in the hot zone and that a refractory of better quality is necessary before this temperature can be increased much further.

ACKNOWLEDGMENTS

This paper is a compilation of the ideas and opinions of too many men in the smelting industry to make individual acknowledgments. Conditions are too complex to admit much certainty in some of the conclusions which have been drawn. The purpose of the paper will have been served, however, if it arouses the interest of smelter men and manufacturers of refractories to the possibilities of increasing furnace capacities by the employment of more suitable refractories.

DISCUSSION

W. B. DONOGHUE, Ottawa, Canada (written discussion).—The cost of fuel in reverberatory smelting constitutes about half of all smelting charges, so in the advance of fuel economy lies a splendid opportunity for metallurgical research and a possible reduction in smelter costs.

During my employment on reverberatory operation I became more and more concerned with the heavy losses of heat due to radiation from the roof and to a less extent from the walls of the furnaces, and I concluded that if this loss could be prevented or controlled a material saving could be effected. The authors refer to "still higher temperatures in the hot zone" but the higher the temperature is in the furnace, the greater is the loss of heat by radiation, especially as the roof becomes thin. In the construction of reverberatories the walls are generally solid walls with no air spaces; the roofs are of solid brick construction, often quite thin and open to free air.

The radiation loss might be improved by constructing a double roof of 8 to 12-in. brick or of sheet metal material, leaving an air space of 12 to 18 in. between it and the roof for heat insulation. It would probably be better to construct the double roof independent of the roof and the temperature of the air in the intervening space could be controlled by drawing off the heated air, which could be used for firing. Cool air could be drawn into the air chamber through vents at the hottest sections or around the slag charging hole and in that way regulate the temperature of the reverberatory roof.

It is difficult to say whether any economy of fuel resulting from a double roof will compensate for the added inconvenience but it appears practical and it would be convenient and not expensive to experiment by constructing a double roof over one

section of a furnace. I am convinced however that in the economy of fuel lies a most attractive possibility for a further reduction in the production cost of copper.

H. A. GRINE, Clarksburg, W. Va.—Has anything ever been done with hydraulic press refractories in connection with the reverberatory furnaces?

G. L. OLDRIGHT.—I do not know that there has. Of course, if the porosity of the brick could be cut down the creeping in of slags could be prevented to some extent. Various kinds of brick have been suggested for the arch—magnesite brick, mullite brick, which is alumina brick, and similar forms. We have referred to the use of a rather thick arch, but what makes brick slag is a great deal of dust at high temperature. The action of a brick can be judged from those two characteristics, together with the composition, but I know of no particular way of making a tight brick by hydraulic pressure.

H. A. GRINE.—There was a hydraulic press refractory put out by the Hydraulic Pressed Brick Co. at St. Louis. I had some experience with this in zinc-furnace lining and it worked out very well. There was much less slag penetration.

G. C. STONE, New York, N. Y.—We have had much experience with arches, but not of that type. The suggestion of covering the arch and making it thicker will not work. It preserves the heat but destroys the arch. We tried that on spelter furnaces. We obtained a beautiful heat but the heat was so great that the arch would not stand. Hard pressed brick will stand better than an ordinary porous brick.

We had one very curious case in which the roof would slag away and for this the silica brick would stand much longer than the clay brick.

S. ROLLE, Carteret, N. J.—I cannot speak for the United Verde Copper Co. but I heard the other day that this company is trying this very thing; it has put in a 30-in. roof made in two layers.

R. M. ROOSEVELT, New York, N. Y.—I think that is common practice around the zinc business, in what we call recuperation. For at least 25 years the arches underneath have been doubled, and the arch has been heated in those chambers when the arch is on top. We have tried a number of different ways, and the recuperation simply takes the heat that goes through the arch and puts it back into the furnace again. It does not keep inside the furnace the heat that is originally placed there. If we get 8 or 10 in. of dust on top of a furnace arch, the arch is going to melt in because the heat has been held in there. The only way the arch can be held is by cooling. This double arch is a recuperation method instead of an insulation method, if I may be allowed to draw the distinction.

S. ROLLE.—I understand the United Verde is doing that very largely because it makes for easy repairs. A very thick roof can be allowed to run longer without repairs than a thin one and when repairs become necessary they are more easily made. The advantages lie in this direction rather than in heat economy.

G. L. OLDRIGHT.—Reference 14 covers the remarks which have just been made; it refers to the same type of roof and the same general idea.

G. E. DALBEY, East Berlin, Conn.—I wonder if there is anything in the following proposition:

As the reverberatory furnaces are constructed and operated, the arch of the reverberatory furnace is in contact with the hottest gas, the slag or products being smelted, in contact with the coldest portion of the gas which is cooled by giving up its heat to the products being smelted. The cool gas, due to the construction of the

furnace, is trapped in the furnace and acts like a blanket covering the products being smelted. The flue gases are taken off at the top of the furnace and the hottest gases are drawn out. If the gas could be taken off at the level of the top of the bath and the cool gases be allowed to flow off and be replaced by hot gas by downward displacement, I believe that it might be possible to greatly increase the thermal efficiency of a reverberatory furnace.

R. M. ROOSEVELT.—I think Mr. Grine has some furnaces which have been in operation for 24 years with the same arches, in which he has that identical thing.

H. A. GRINE.—The stack opening is low down. Those arches have been in for over 24 years.

R. M. ROOSEVELT.—The middle walls have been rebuilt, if I remember correctly, a number of times, but the arches still hang there, and the temperature inside the furnaces gets up to approximately 1300° C. I do not know of any other furnace that has done so well.

C. S. WITHERELL, New York, N. Y.—Will Mr. Oldright tell us something about the effect of the mortar used for making the joints of the brickwork upon the slagging thereof?

G. L. OLDRIGHT.—In laying the brick arches of reverberatory furnaces, little, if any, grouting is used at present. The regular practice is to lay the bricks dry and to sweep a little finely ground silica over their tops to fill in the cracks between them to some extent. The users of brick also specify that the brick should be straight and true, and care is taken to fit the brick together as closely as possible to lessen the amount of air leakage. In earlier practice, it was sometimes the custom to dip the brick in a thin grouting of fireclay. Such a procedure is now thought to have lessened the life of the arch, but owing to changes in practice it would be difficult to ascribe accurately any very definite figure to this decrease.

Our reference No. 15 gives experimental data on the lowering of the fusion point of pure silica by the addition of various bases. One might add to this reference a note that the work was continued by the author, Mr. Grieg, in the issue of the same journal for December, 1927. As a summary, one might say roughly, that the presence of a small amount of lime in the grouting is not so very harmful. The presence of alumina or of the oxides of sodium or potassium is bad, as it lowers the fusion point of pure silica very greatly. (Cristobalite, which is formed from silica at high temperature, melts at about 1710° C.) The presence of ferric oxide is much less harmful, if the oxides of sodium and potassium are not present. With alumina, the curve of temperature versus fusion point drops almost straight down from 1710° as alumina is added; in the case of the addition of iron the fusion point also drops greatly, while for potassium, if enough were added it would be possible to melt the silica at a few hundred degrees Centigrade.

R. P. HEUER, Philadelphia, Pa.—What other refractories were considered for the service outlined in this paper? Mr. Oldright referred to mullite. Probably we could obtain refractories which would be better than silica brick but they might cost 20 times as much. In this particular case, would not the cure be worse than the disease? Our present silica brick have a fusion point of about 1700° C. They retain their structural strength upon heating to within a very few degrees of their actual melting point. The recent work of Grieg¹⁷ has shown that silica is remarkably resistant to the corrosive attack of FeO and similar oxides due to the phenomenon of immiscibility of the liquid slags formed. This paper shows that silica brick have

¹⁷ U. W. Grieg: *Op. cit.*

caused the failure of a furnace roof at 1430° C. The operation of this furnace was attended by the impingement upon the roof of flame and dust arising from the ore charge. Perhaps the solution of the problem is as much an engineering matter as it is a matter of refractories.

G. L. OLDRIGHT.—In the paper we said that the smelter men tell us “the brick are pretty cheap and mighty good, and give some mighty long lives.” The object of this paper is to show the refractory makers, and also the users of brick, that if we can get a brick that will stand higher temperatures, we can increase the intensity of combustion and thereby raise the temperature for smelting and lower costs. If I knew what particular brick would do this, I should be very glad to reveal it.

I have given some general suggestions on cutting down the amount of dusting, but the diminution of dusting will only increase the life of the brick at present operating temperatures. I doubt very greatly, unless the dust could be absolutely cut out instead of only being diminished, whether the use of very much higher operating temperature would be permissible, because the causes that have been described by which the slag penetrates up through the brick would still hold, and of course with a higher temperature less base is needed to satisfy the acid.

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Formation and Decomposition of Zinc Ferrite*

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METALLURGISTS differ considerably in their opinions regarding the effect, if any, of small amounts of iron pyrites, or other iron compounds on zinc sulfide ores during the roasting operation. As a result, the literature on this subject expresses many conflicting ideas.

The ore, previously crushed to pass through a 1-mm. screen, dries in the roasting kiln, losing water up to 200° to 300° C. At 150° C. iron pyrites begin to lose sulfur. As the temperature increases, while the ore is passing to the hotter part of the furnace, the sulfur burns to sulfur dioxide, leaving the metals as zinc oxide and ferric oxide. These two substances then interact to form zinc ferrite, a compound, yellow brown in color, having the general characteristics of a spinel. Like magnetite, another spinel, zinc ferrite is insoluble in alkalis and dilute acids. As a result, when leached with 10 per cent. sulfuric acid solution, zinc values tied up in this compound remain behind.

HISTORY

In the electrolytic recovery of zinc from its ores, the roasted ores are usually leached with a 10 per cent. sulfuric acid solution. When the zinc ore contains iron pyrites as an impurity in an appreciable amount, the actual yield of metallic zinc is much lower than a theoretical calculation would indicate. Further, losses of zinc are, in a rough way, directly proportional to the amount of iron pyrites or other iron compounds present. When the iron is present in the ore in an isomorphous relation to the zinc, as in the case of a marmatite, a greater amount of zinc remains insoluble than when present as a true pyrite or marcasite. Work on the insoluble residues obtained from leaching of the calcines shows that the zinc lost is intimately tied up in this residue. Some investigators have suggested a definite compound, such as zinc ferrite; others merely a mixture of zinc and iron compounds.

As early as 1851, Pelouze⁽¹⁾¹ prepared calcium ferrite by fusion of calcium oxide and ferric oxide. Ebelman,⁽²⁾ about the same time, prepared

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¹ Numbers in parentheses refer to bibliography on last page.

zinc ferrite in the same manner. More recently, Hofman^(3,4) has made a study of zinc-roasting problems, using zinc oxide and ferrous sulfate, zinc oxide and ferric oxide, and zinc ores containing iron compounds. After the samples were heated, in a porcelain crucible, with a Bunsen burner, and were cooled, the mixture was leached with ammonium sesquicarbonate. The amount of zinc oxide remaining in the insoluble residue seemed to indicate that no definite compound was formed. Hofman found that the time of heating need not be greater than 1 to 2 hr. His conclusion that no definite compound was formed was probably due to the fact that the oxides were neither ground to sufficient fineness nor given sufficient chance to become intimately mixed.

In 1908, Wells⁽⁵⁾ prepared zinc ferrite by evaporating to dryness a solution of zinc sulfate containing iron salts in varying proportions, and heating this residue to a temperature of 800° to 900° C. for 2 hr. The zinc oxide unaffected by the reaction was leached out with Low's reagent.² The insoluble material was shown to contain zinc oxide and ferric oxide in the proportion represented by the formula $\text{Zn}(\text{FeO}_2)_2$. Wells concluded that in the roasting of zinc ores containing iron pyrites it is practically impossible to prevent the formation of zinc ferrite.

The following year Hilpert⁽⁶⁾ examined a number of compounds having the general formula, $\text{M}''\text{O} \cdot \text{Fe}_2\text{O}_3$ including zinc ferrite. The ferrites were prepared by the fusion of the requisite oxides.

In 1912, using ferruginous zinc ores, Hommel⁽⁷⁾ came to the conclusion that the formation of ferrites was due to faulty working of the ore during roasting.

The most recent work on this subject was done by Hamilton, Murray and McIntosh.⁽⁸⁾ They found that zinc oxide and iron oxide combined at temperatures above 649° C. in the proportion of one molecule of zinc oxide to one of iron oxide.

These references show that opinions are conflicting as to the possible formation of a compound between zinc oxide and iron oxide, and that there is considerable doubt concerning the composition of such a compound even if one is formed, as well as concerning its temperature of formation.

MATERIALS AND APPARATUS USED

The ferric oxide and zinc oxide used in these experiments were chemically pure. The magnesium oxide was U. S. P. The calcium hydroxide was prepared from burnt marble by slacking and drying. The zinc ore, from southwestern Wisconsin, was obtained from the Department of Metallurgical Engineering of the University of Wisconsin.

² Low's reagent consists of a solution of 200 g. of ammonium chloride in 500 c.c. of ammonium hydroxide (sp. gr. 0.9) and 750 c.c. of water.

The materials were all ground and passed through 100-mesh screens. The samples then were mixed by rotating for a number of hours in a bottle about 4 in. in diameter and 8 in. high. The bottle was fastened between two disks so that it could rotate about its longer axis.

The samples were heated in fused silica trays in an electric muffle furnace of the resistance type which had a built-in rheostat for temperature control. The temperature of the furnace was determined by means of a chromel-alumel thermocouple.

FORMATION OF FERRITES

ZINC FERRITE

Zinc oxide and ferric oxide were separately passed through a 100-mesh sieve. Varying amounts of these oxides were thoroughly mixed in the rotary mixer, to get intimate contact between the particles. The mixtures were then heated at various temperatures and for various lengths of time in the electric muffle furnace. After this heating the samples were analyzed for ferric oxide and soluble and insoluble zinc oxide. Soluble zinc oxide is that which can be extracted with Low's reagent.

Influence of Variation in Concentration of Zinc Oxide on Formation of Zinc Ferrite

Various amounts of the sifted zinc oxide and ferric oxide were weighed out and mixed in the rotary mixer for 12 hr. Samples were then heated in the muffle for 12 hr. at 850° C.

A sample of the calcined mixture weighing one gram was extracted with Low's reagent. The soluble portion was analyzed for zinc. Zinc and iron were determined in the residue.

Before heating, the mixture was 100 mesh in size. Its color was that of ferric oxide, a distinct red. After heating, the mixture was yellow brown in color and was slightly coarser than 100 mesh. The mass did not fuse at these low temperatures but remained in the granular condition. All of these experiments were carried out at temperatures several hundred degrees below the melting point of any one of the constituents.

The results of this series of experiments are shown in Table 1.

Since the analysis of the samples showed the residue insoluble in Low's reagent to be composed of one mole of zinc oxide to one mole of ferric oxide, the zinc ferrite formed was assumed to have the formula $\text{Zn}(\text{FeO}_2)_2$. A compound of this composition was always formed when zinc oxide and iron oxide were heated at 850° C. for 12 hr., regardless of the proportion of the oxides used.

TABLE 1.—*Formation of Zinc Ferrite by Using Various Proportions of Zinc Oxide and Ferric Oxide*

Sample Number	Proportions Used (Moles)		Analysis			Molal Ratio in Residue	
	ZnO	Fe ₂ O ₃	ZnO Soluble, Grams	ZnO Insoluble, Grams	Fe ₂ O ₃ , Grams	ZnO	Fe ₂ O ₃
1	½	1	none				
2	4	1	0.4827	0.1649	0.335	1	1.05
3	32	1	0.9137	0.030	0.058	1	0.99
4	256	1	0.9094	0.0293	0.0559	1	0.97
5	400	1	0.9672	0.0101	0.0202	1	1.01

Influence of Variation in Heating Temperature on Formation of Zinc Ferrite

In this series of experiments sifted zinc oxide and ferric oxide were weighed out in proportion of 64 moles³ of zinc oxide to one of ferric oxide, then thoroughly mixed in the rotary mixer. Portions were heated for 3 to 4 hr. in the muffle at temperatures varying from 540° to 1000° C. The material did not undergo any change until a temperature of 650° C. was reached, when there was a color change to a yellow brown. Up to this temperature, qualitative examination showed no insoluble zinc oxide, so it was assumed that there was no reaction below this temperature. From 650° to 1000° C., the mixture did not undergo further change in color or general appearance and all samples showed insoluble zinc oxide. The last sample of the series was analyzed and the insoluble portion was found to contain zinc oxide and ferric oxide in the proportion of one mole

TABLE 2.—*Formation of Zinc Ferrite at Various Heating Temperatures*

Sample Number	Temperature of Heating, Degrees C.	Time of Heating, Hours	ZnO Soluble, Grams	ZnO Insoluble, Grams	Fe ₂ O ₃ , Grams	Molal Ratio in Residue	
						ZnO	Fe ₂ O ₃
1	560	4		none			
2	600	4		none			
3	640	3		none			
4	650	2	0.9288	0.0315	0.0614	1.00	0.99
5	750	2					
6	900	2					
7	925	2					
8	975	2	0.9214	0.0302	0.0600	1.00	1.01

³ The proportion of one mole of ferric oxide to 64 moles of zinc oxide was selected in order to insure the presence of sufficient zinc oxide in contact with the iron oxide to cause a complete reaction to take place with respect to ferric oxide present.

to one mole, showing that zinc ferrite was formed and that its composition was independent of its temperature of formation through this temperature range.

Table 2 shows the results of these experiments. In each case a one-gram sample was used for analysis.

Influence of Variation in Time of Heating on Formation of Zinc Ferrite

Zinc oxide and ferric oxide, previously sifted through 100 mesh, were weighed out in the proportion of one mole of ferric oxide to 64 moles of zinc oxide and mixed thoroughly in the rotary mixer. After mixing, a sample was analyzed qualitatively for insoluble zinc oxide and found to contain none. The mixture was then put in the electric muffle furnace at 850° C. The temperature was held constant and samples were removed at intervals up to 100 hours.

At the end of one hour a sample was removed. No zinc oxide was found in the insoluble portion, which was probably due to the fact that the furnace was cooled down by introducing a large mass of cold material so that it did not reach the reaction temperature again by the end of the first hour. At the end of the second hour a sample of the mixture was removed. Extraction of the mixture was made with Low's reagent, and the residue from this leaching was analyzed for zinc and ferric oxides as before. The molal ratio between these two was again found to be one to one. The last sample of the series, which was in the muffle furnace for 96 hr., was analyzed and found to be of the same composition as the sample taken at the end of the 2-hr. period. The color of these two samples, as well as their analysis, was the same, and since the color of the intermediate samples was also the same, the intermediate samples were assumed to be of the same composition and were not analyzed. The experiments show the composition of zinc ferrite to be uninfluenced by length of heating between the time limits of the experiment.

Table 3 gives the results of this series of experiments. In each case a one-gram sample was used for analysis.

TABLE 3.—*Formation of Zinc Ferrite, with Various Lengths of Heating Periods*

Sample Number	Temperature of Heating, Degrees C.	Time of Heating, Hours	ZnO Soluble, Grams	ZnO Insoluble, Grams	Fe ₂ O ₃ , Grams	Molal Ratio in Residue	
						ZnO	Fe ₂ O ₃
1	0	0		none			
2	850	1		none			
3	850	2	0.9210	0.0284	0.0555	1.00	0.99
4	850	96	0.9219	0.0283	0.0555	1.00	0.99

CALCIUM FERRITE

Since ferric oxide and zinc oxide react to form a definite compound—zinc ferrite—at temperatures below their fusion temperature, it was thought that possibly calcium oxide and ferric oxide would interact at a low temperature also, since calcium ferrite had previously been prepared by fusion of the oxides. If calcium oxide and ferric oxide did form a compound calcium ferrite, this compound might also be formed by the interaction of zinc ferrite and calcium oxide, forming free zinc oxide at the same time. Zinc oxide could then be leached out in the ordinary metallurgical process.

To determine whether or not this reaction between calcium oxide and ferric oxide did take place in the dry state below the fusion temperature, ferric oxide and calcium hydroxide, screened as previously described, were mixed intimately in the proportion of 1 mole of ferric oxide to 2 moles of calcium hydroxide. Portions of this mixture were heated at temperatures varying from 500° to 950° C. for periods of about 12 hr. At 900° C. there was a color change from a light to a much darker red. A number of experiments were made at this point to find a reagent suitable for leaching free calcium oxide away from that combined as calcium ferrite. Solutions of acetic acid varying from 1 to 5 per cent. were prepared. A portion of each solution was used to treat samples of calcium oxide, calcium ferrite, calcium carbonate and ferric oxide. Each solution was brought to the boiling point and held there for from 10 to 20 min. The calcium carbonate and calcium oxide were not readily soluble except in the 4 and 5 per cent. solutions. The ferric oxide was insoluble in all of the solutions. The calcium ferrite seemed to be unattacked by the reagent. Consequently, 4 per cent. acetic acid was selected as the reagent for dissolving free calcium oxide.

TABLE 4.—*Calcium Ferrite*

Sample Number	Number of Extractions	Insoluble CaO, Grams	Fe ₂ O ₃ , Grams	Molal Ratio in Residue	
				CaO	Fe ₂ O ₃
1	1 (below b.p.)	0.2014	0.7030	1.00	1.22
2	1 (below b.p.)	0.2012	0.6941	1.00	1.24
3	1 (at b.p.)	0.1152	0.6956	1.00	2.12
4	1 (at b.p.)	0.1152	0.6956	1.00	2.12
5	2 (at b.p.)	0.0624	0.6674	1.00	3.76
6	2 (at b.p.)	0.0688	0.6674	1.00	3.34
7	3 (at b.p.)	trace			
8	3 (at b.p.)	trace			

The sample which had been heated at 900° C. was then analyzed quantitatively, using 4 per cent. acetic acid for extraction. The soluble and insoluble portions were analyzed, the results showing that if a compound is formed it is unstable toward acetic acid. (See Table 4.)

MAGNESIUM FERRITE

Magnesium ferrite was formed in a manner analogous to the method used in the formation of zinc ferrite. Screened magnesium oxide and ferric oxide were weighed out in proportion of one mole of ferric oxide to 24 moles of magnesium oxide, heated in the muffle, and analyzed. Analy-

TABLE 5.—*Magnesium Ferrite*

Sample Number	Proportions Used (Moles)		Temperature of Heating, Degrees C.	Time of Heating, Hours	Analysis Insoluble		Ratio in Insoluble Portion	
	MgO	Fe ₂ O ₃			MgO, Grams	Fe ₂ O ₃ , Grams	MgO	Fe ₂ O ₃
1	24	1	700	18	0.0424	0.1692	1.00	1.01
2	24	1	975	12	0.0409	0.1711	1.00	1.05

sis of these samples was carried out in practically the same manner as that of the samples analyzed in connection with the formation of zinc ferrite. The results, using one-gram samples, are shown in Table 5.

DECOMPOSITION OF ZINC FERRITE

It has been shown that when zinc oxide and ferric oxide are in intimate contact with one another at or above 650° C., a compound is formed. If maximum extraction of zinc from roasted ores is to be had, it is necessary to decompose any zinc ferrite that may form. When magnesium oxide, and probably when calcium oxide, is heated with ferric oxide a reaction takes place; therefore, it was thought that a substitution might take place, if calcium oxide or magnesium oxide were heated with zinc ferrite. The theory was that the zinc oxide combined in the zinc ferrite would be freed by the substitution of calcium or magnesium oxide, and that the zinc oxide could be leached out of the mixture leaving the calcium or magnesium ferrite insoluble in Low's reagent.

Thus, in roasting ores containing zinc and iron compounds, the addition of magnesium oxide would prevent the formation of zinc ferrite and the yield of zinc would approach more nearly the theoretical.

Introduction of Calcium Oxide

In this series of experiments, two compounds of calcium which would yield the oxide on heating were used; namely, calcium carbonate and calcium hydroxide.

Zinc ferrite and calcium carbonate were screened through 100-mesh screens, mixed thoroughly and heated in the muffle at 850° C. for 12 hr. When they were removed from the furnace, it was found that the color had changed from a yellow brown to a dull grayish black. Qualitative examination showed that all of the calcium carbonate had not decomposed. In view of this fact, it was thought best to use calcium hydroxide in the decomposition reactions rather than calcium carbonate, as more calcium would thus be available for the reaction.

Calcium hydroxide was accordingly substituted for the carbonate in the above procedure. After heating at 950° C. for 100 hr., much the same color change had taken place. Closer inspection revealed small yellow brown specks throughout the mass, similar in color to the original mixture. This color change indicated that a reaction had taken place.

Qualitative analysis showed that some of the zinc oxide, which before roasting made up the zinc ferrite, could now be leached out either with Low's reagent or 4 per cent. acetic acid solution. Quantitative determinations, as shown in Table 6, were made.

TABLE 6.—*Analysis of Samples of Zinc Ferrite Heated with Calcium Oxide*

Sample Number	Proportion Used, Grams		Soluble ZnO, Grams	Soluble CaO, Grams	Insoluble CaO, Grams	Insoluble ZnO, Grams	Fe ₂ O ₃ , Grams	Zn Recovered, Per Cent.
	CaO	Zn(FeO ₂) ₂						
1	50	105	0.0548	0.2328	trace	0.1781	0.5255	23.5
2	50	105	0.0580	0.2344	trace	0.1679	0.5301	25.7
3	50	50	0.1614		trace	0.0246	0.3760	86.8
4	50	50	0.1639		trace	0.0245	0.3713	87.0

In the last column, headed "Zn recovered, per cent.," is shown the percentage of zinc oxide that was freed from zinc ferrite by calcium oxide. This is found by dividing the soluble zinc by soluble zinc plus insoluble zinc.

In the column headed "Proportions Used" are given the amounts of material that were weighed out roughly before mixing. It will be noted that in all cases an excess of calcium oxide over that necessary for the displacement action was used. For instance, in the first case 50 g. of calcium oxide were used to 105 g. of zinc ferrite. This is roughly two moles of calcium oxide to one mole of zinc oxide combined as zinc ferrite. This procedure was followed in order to make the reaction more complete if possible. In reactions of this type, no reaction will take place unless the reacting substances are in actual contact with one another.

Introduction of Magnesium Oxide

Magnesium oxide and zinc ferrite ground to 100 mesh were mixed thoroughly and heated in an electric muffle. No color change was noted after heating. Analysis showed that some of the zinc oxide, which before heating made up zinc ferrite, was now free and could be leached out with Low's reagent, whereas some of the magnesium oxide, all of which was originally soluble in Low's reagent, was now insoluble. Analysis showed that one molecule of zinc oxide was replaced in this reaction by one molecule of magnesium oxide. However, the reaction was by no means complete, as can be seen in Table 7.

In addition to the above-described samples made up from pure materials, a sample of zinc ore was also studied in relation to the increase in recovery of zinc by addition of magnesium oxide. The ore, a sphalerite containing 40 to 50 per cent. of iron pyrites, was ground to 100 mesh and roasted in the electric muffle. When it was placed in the furnace, the temperature was 400° C. A temperature of 400° to 500° C. was maintained for 24 hr. and then raised gradually to a maximum of 950° C., where it was held for 100 hr. A second sample thoroughly mixed with an equal weight of magnesium oxide was treated as nearly as possible in the same manner. These mixtures were analyzed by the method used with zinc ferrite and calcium oxide. The results of the experiments are shown in Table 9.

TABLE 7.—*Effect of Magnesium Oxide on Recovery of Zinc*

Sample Number	Proportion Used, Grams		Insoluble ZnO, Grams	Insoluble MgO, Grams	Fe ₂ O ₃ , Grams	Total (Calculated) ZnO	Per Cent. Replacement
	MgO	Zn(FeO) ₂					
1	10	50	0.1849	0.0576	0.5698	0.3031	38.6
2	17	50	0.2056	0.0386	0.5518	0.2836	27.5
3	50	50	0.1268	0.0381	0.3948	0.2038	37.8
4	125	50	0.0951	0.0271	0.2914	0.1498	36.5

In Table 7, the column headed "Total ZnO (calculated)" was calculated by finding the amount of zinc oxide that is chemically equivalent to the insoluble magnesium oxide. To this was added the insoluble zinc oxide. We thus have the amount of zinc oxide originally combined with the ferric oxide.

In addition to mixtures of magnesium oxide and zinc ferrite, a mixture of zinc oxide, ferric oxide and magnesium oxide was heated. The zinc oxide and ferric oxide were present in the proportion of 1 mole to 1 mole, so that we have in reality a mixture which on heating will give zinc ferrite. In addition, the magnesium oxide was added in the proportion of 1 mole of zinc oxide to 2 moles of magnesium oxide. In other

words, this sample was the same as Sample 2 of Table 7. There is more soluble zinc oxide in this case than when magnesium oxide was heated with zinc ferrite; this is probably due to better mixing and better contact in case of the three oxides than where zinc ferrite was heated with magnesium oxide. Consequently there is more chance for a reaction to take place.

Table 8 shows the results of heating a mixture of zinc oxide, magnesium oxide and ferric oxide at 900° C.

TABLE 8.—*Results of Heating a Mixture of Three Oxides*

Proportions Used, Grams			Tempera- ture, Degrees C.	Insoluble ZnO, Grams	Insoluble MgO, Grams	Fe ₂ O ₃ , Grams	Total Calculated, ZnO	Per Cent. Replace- ment
MgO	ZnO	Fe ₂ O ₃						
17	17	33	900	0.1531	0.0549	0.5226	0.264	41.6

TABLE 9.—*Results of Roasting Samples of Zinc Ore*

	Insoluble ZnO, Grams	Fe ₂ O ₃ , Grams	Soluble ZnO, Grams	Per Cent. Recovery
Raw ore.....	0.0382	0.1293	0.0758	66.5
Ore and MgO.....	0.0248	0.1283	0.0918	78.7

Table 9 shows the results of experiments on the zinc ore. Sample 1 consisted of the raw ore alone. Sample 2 consisted of the raw ore roasted with an equal weight of magnesium oxide. Since the quantities of ore are the same in each sample, the percentage replacement of zinc oxide by magnesium oxide in the insoluble will be shown by the decrease of insoluble zinc. Thus we have

$$\frac{0.0382 - 0.0248}{0.0382} = 35 \text{ per cent.}$$

Heating Zinc Ferrite under Pressure with Sodium Hydroxide Solution

Another possible method of decomposition of zinc ferrite suggested itself. It was thought that zinc ferrite might be somewhat soluble in sodium hydroxide solution under pressure. If this were the case the zinc ferrite would dissolve slightly in sodium hydroxide solution, with which it would immediately react to form ferric hydroxide, which is insoluble, leaving sodium zincate in solution.

A sample of zinc ferrite was suspended in a 25 per cent. sodium hydroxide solution in an autoclave and heated 2 hr. at 70 lb. pressure (150° C.). After cooling, the solution was analyzed qualitatively for zinc, which was not found; therefore no zinc ferrite had been decomposed.

METHODS OF ANALYSIS

CHEMICAL METHODS

In looking over the literature, it was found that practically all of the investigators had used Low's reagent as an extracting agent for so-called soluble zinc. Hofman recommended ammonium sesquicarbonate as an extracting agent superior to others. Preliminary experiments described in this paper showed that Low's reagent was satisfactory, hence it was used throughout the investigation.

Since free calcium oxide could not be dissolved in Low's reagent, this solvent could not be used where free or combined calcium oxide was being determined. In this case a 4 or 5 per cent. acetic acid solution was tried, as described above. One objection to this reagent was that it formed a colloidal solution very easily with the materials treated. Often this gave considerable trouble, by necessitating refiltration or discarding of the sample because of inability to wash the precipitate sufficiently to remove mechanically occluded soluble matter.

The following procedure was found more satisfactory in dealing with the acetic acid extraction, as a satisfactory separation of soluble from insoluble matter could be secured. After heating the sample just to the boiling point with acetic acid containing 5 per cent. ammonium chloride, the suspended matter was allowed to settle. The liquid was decanted off as completely as possible. A second extraction was made in the same manner, and the liquid added to the first extraction. This solution was then evaporated to one-half its original volume. The solid matter was again treated with 4 per cent. acetic acid solution and a few grams of solid ammonium chloride added. After boiling 5 to 15 min., the solution was filtered through a Gooch crucible, through which the first extractions had been filtered. By this method, a clear filtrate was always obtained.

When free magnesium oxide was present, it could be extracted with the soluble zinc by the use of Low's reagent.

X-RAY SPECTROGRAPH METHOD

While the methods of chemical analysis described above seemed to indicate that definite compounds were formed in the cases indicated, chemical analysis cannot be considered absolute proof. Certain physical constants should be studied which will differentiate, in themselves, between the constituents that go to make up the supposed compound and the compound itself.

In the case of ferrites, one property that immediately suggested itself was the arrangement of the atoms, which is different in every group of compounds. This property is made use of in X-ray spectrography, which

shows the atomic arrangement or space lattice as a definite pattern on the photographic plate.

Accordingly, X-ray spectrographs were made of the materials used in these experiments. Since every group of compounds has its own type of spectrographic pattern, the oxides making up the ferrites should have patterns different from the patterns of the ferrites, if such a compound was present. If a ferrite did not form, we should get a pattern made up of the combined patterns of the two oxides. Examinations of the X-ray spectrographs taken of the insoluble residue from heating zinc oxide and ferric oxide showed a pattern entirely different from that of either zinc oxide or ferric oxide. Hence a new compound zinc ferrite had formed. Since the old lines of zinc oxide and ferric oxide patterns were absent, it showed that, with the exception of amounts of impurities so small as not to be detected, the only thing in the insoluble residue was a compound of zinc ferrite; that is, there was no uncombined ferric oxide. The same thing was true of the residues from heating calcium oxide and ferric oxide, magnesium oxide and ferric oxide. Occasionally, very faint lines of an oxide pattern were observed, showing that all of that particular oxide had not been extracted.

In the case of zinc ferrite and magnesium ferrite, we have a typical spinel pattern. In the case of calcium ferrite, we have a different pattern, showing that calcium ferrite does not belong to the spinels. Since the chemical analysis for the various insoluble residues obtained agree within themselves, and since we have a spectrograph pattern differing from the pattern of the constituents, it may be definitely and positively stated that compounds have been synthesized which are magnesium ferrite, $\text{Mg}(\text{FeO}_2)_2$, and zinc ferrite, $\text{Zn}(\text{FeO}_2)_2$.

Discussion of the X-ray Spectrograph Plates

The lines shown at the left end of the pattern are made by light rays of the least angle of reflection. They are much more distinct but at the same time less characteristic because of the small angle of deflection and consequent closeness of lines of different patterns. The lines in the middle and right end of the plates are made by light rays with greater angle of reflection. These lines are much less distinct and at the same time more characteristic of the substance. For instance, a superficial examination of the patterns of zinc ferrite, Fig. 2, and magnesium ferrite, Fig. 3, will show at first no difference in the patterns, as they are both spinels and consequently show the spinel pattern. Closer examination, however, will show a difference, not in number of lines or spacing arrangement but in the distance between the lines. This is more noticeable at the right end of the plates. It will be seen that the lines are farther apart in the magnesium ferrite than in the zinc ferrite pattern. This is inter-

Fig. 1.

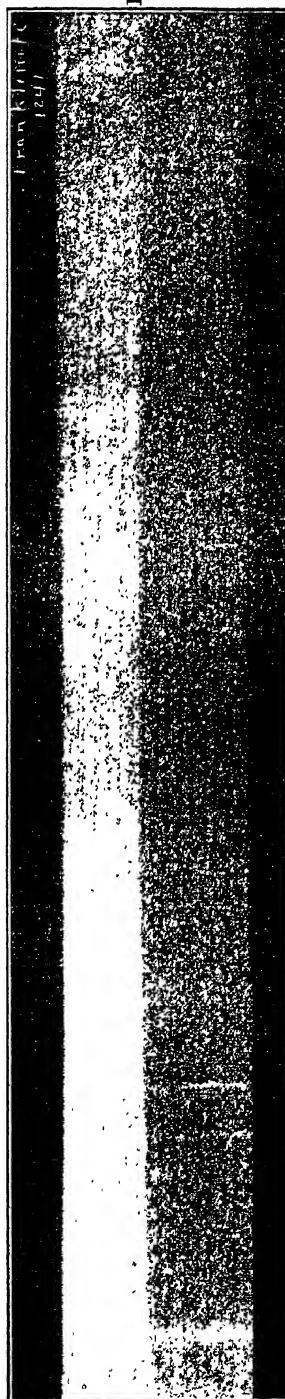


Fig. 2.

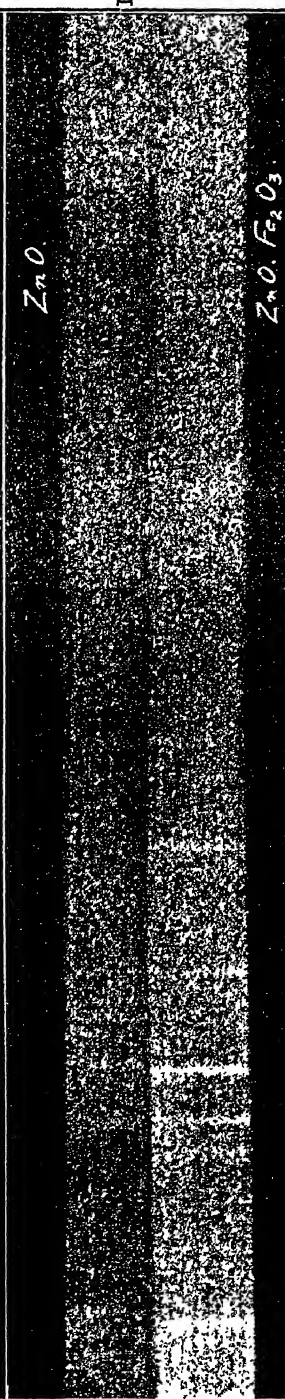
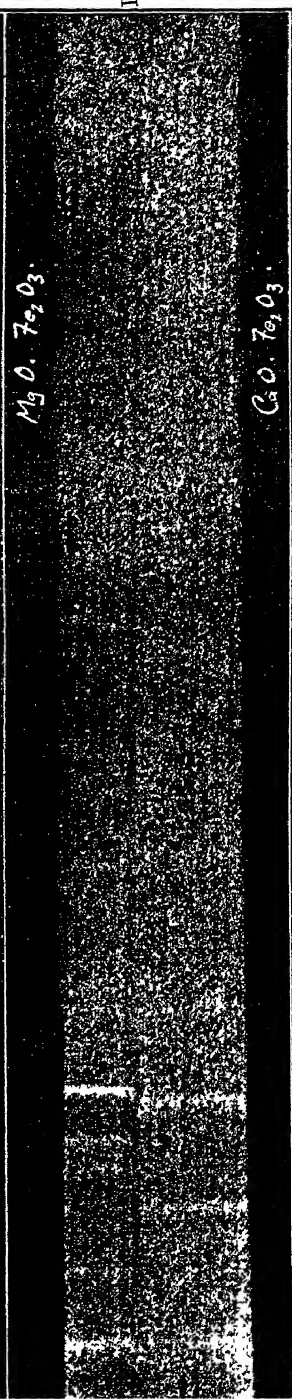


Fig. 3.



FIGS. 1-3.—X-RAY DIFFRACTION SPECTRA OF FRANKLINITE, ZINC OXIDE, AND ZINC, MAGNESIUM AND CALCIUM FERRITES.

FIG. 4.

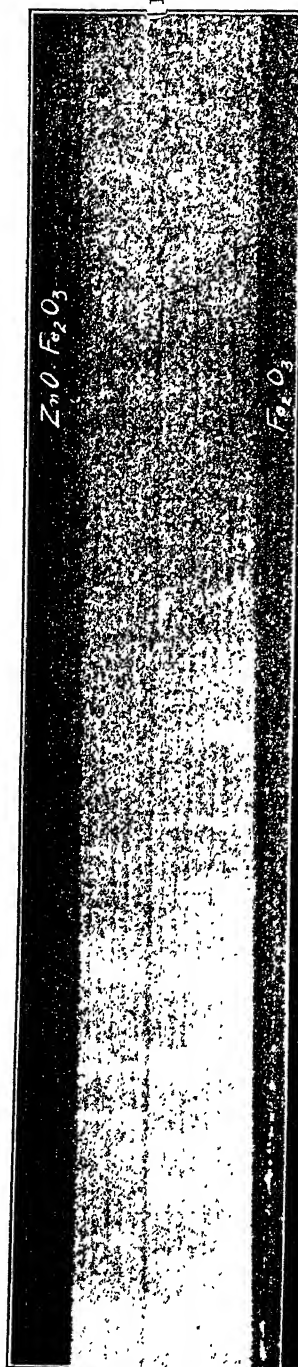


FIG. 5.

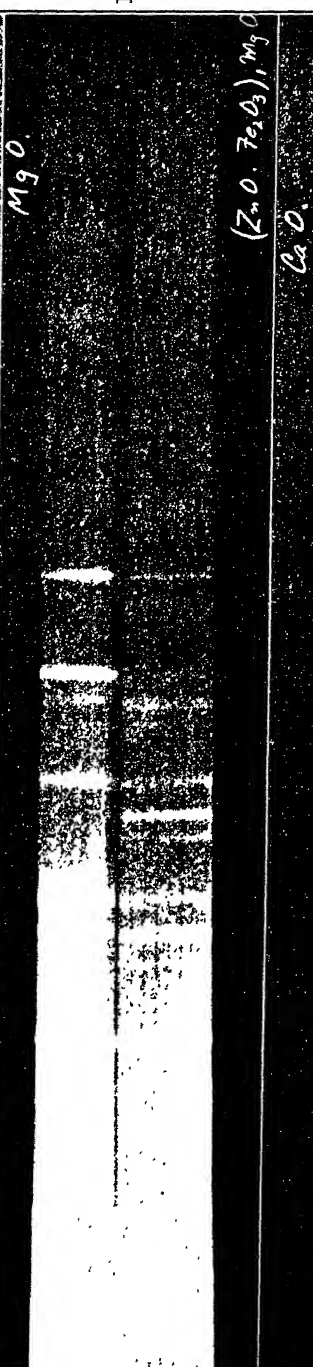
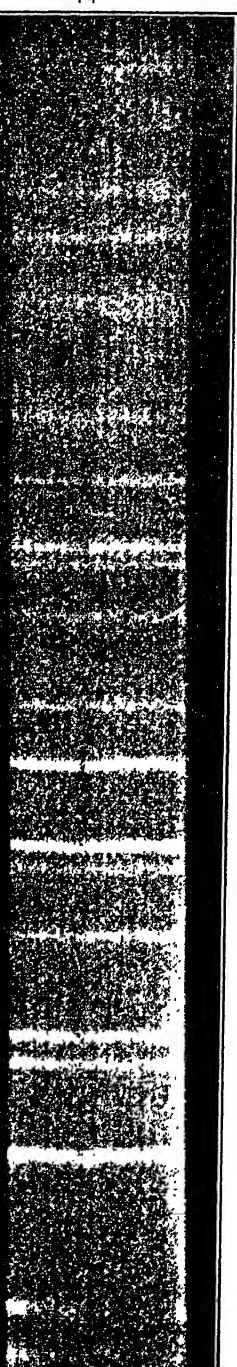


FIG. 6.



FIGS. 4-6.—X-RAY DIFFRACTION SPECTRA OF ZINC FERRITE, FERRIC OXIDE, MAGNESIUM OXIDE, ZINC FERRITE WITH MAGNESIUM OXIDE, AND CALCIUM OXIDE.

preted as meaning that in magnesium ferrite the magnesium atom occupies slightly less space in the space lattice than does zinc in the zinc ferrite lattice.

In Fig. 1, we have the pattern of the mineral Franklinite, the natural zinc ferrite. Franklinite is a spinel. The spectrograph shows the characteristic spinel pattern. Comparing this pattern with the artificial Franklinite or zinc ferrite, Fig. 2, prepared in these experiments, we see that, as far as can be observed, the patterns are identical. Magnesium ferrite, as mentioned above, is also a spinel showing the same characteristic spinel arrangement. On the other hand we have calcium ferrite (Fig. 3), a compound similar from a chemical standpoint, having a pattern of an entirely different type. Calcium ferrite is therefore not a spinel. The explanation at present offered is that the calcium atom, having a volume of 21 cubic Ångströms, is entirely too large to go into the spinel space lattice.

If the patterns of the various compounds that have been synthesized are compared with the patterns of the constituents used in their synthesis, they will be found to be absolutely different. For instance, compare the pattern of zinc ferrite, Fig. 2, with the patterns of the materials from which it was synthesized—zinc oxide, Fig. 2, and ferric oxide, Fig. 4. Also compare magnesium ferrite, Fig. 3, with magnesium oxide, Fig. 5, and ferric oxide; and calcium ferrite, Fig. 3, with calcium oxide, Fig. 6, and ferric oxide. This shows that new compounds have been formed.

Fig. 7 shows a comparison of the zinc ferrite pattern with the pattern made by a mixture of ferric oxide with an excess of zinc oxide, which has been heated but from which the soluble zinc oxide has not been extracted.

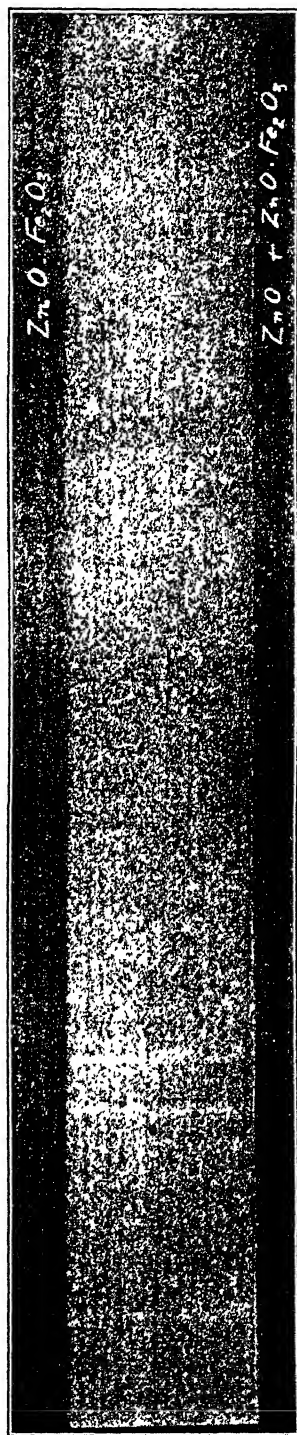
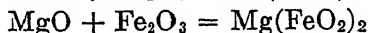
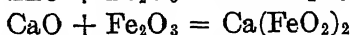
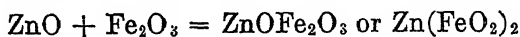


Fig. 7.—X-RAY DIFFRACTION SPECTRA OF PURE ZINC FERRITE, AND ZINC FERRITE WITH EXCESS ZINC OXIDE.

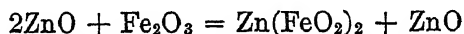
This gives a pattern of zinc ferrite and also a pattern of zinc oxide caused by the excess of zinc oxide.

GENERAL DISCUSSION

When two oxides of different degrees of basicity or acidity are brought into intimate contact at or above their initial reaction temperature, a reaction will take place, forming a salt. One of the oxides acts as a base and the other as an acid, much as in the ordinary type of aqueous neutralization reaction, except that no solvent enters into the reaction. We have then a basic oxide, such as zinc oxide or magnesium oxide, reacting with a less basic or more acidic oxide, ferric oxide, to form a salt:



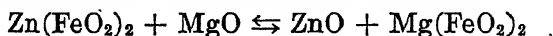
If an excess of zinc oxide, calcium oxide, or magnesium oxide is present, the oxide remains free in the mixture and a more basic compound does not form.



No reaction appeared to take place between zinc oxide and ferric oxide at temperatures below 650° C. When the reaction did take place there was a color change; also a slight enlargement in particle size was noted.

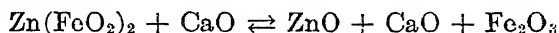
The rate of reaction appeared to be quite rapid after the initial temperature of reaction was reached.

In decomposing zinc ferrite, two entirely different types of reaction appeared to take place, depending on whether calcium oxide or magnesium oxide was used as decomposing agent. When magnesium oxide was used, the following equilibrium reaction occurred:



Analysis showed that before the mixture of zinc ferrite and magnesium oxide was roasted, all of the magnesium was soluble. After roasting, part of the zinc was soluble and part of the magnesium was insoluble. Quantitative examination showed that for every molecule of zinc oxide rendered soluble, one molecule of magnesium oxide became insoluble. This shows that a simple replacement reaction occurred, which would be expected from present theories of crystal structure. The zinc atom and the magnesium atom are practically the same size, zinc being 9.7 cubic Ångströms in volume and magnesium being 12. Hence these atoms can replace each other very easily without disrupting or distorting the space lattice. The X-ray spectrograph shows zinc ferrite and magnesium ferrite to be almost identical. Hence we would expect the simple replacement reaction which is observed.

When calcium oxide was used as the decomposing agent, the following reaction apparently occurred:



That this reaction did occur is difficult to prove, as the decomposition was in no case complete. There was a color change from a light brown to a dull gray black. Since calcium ferrite is a red brown in color, its formation would not account for the color. A compound of zinc oxide and calcium oxide, calcium zincate, is white. Since there is no calcium in the insoluble portion of these samples after extraction with acetic acid solution, and since part of the zinc is leached out by the acetic acid, the only other explanation is to consider that free ferric oxide is formed in the reaction. According to Mellor,⁽¹⁰⁾ the color of ferric oxide depends largely on its temperature of formation and its state of subdivision. When zinc ferrite decomposes, the ferric oxide resulting would quite naturally be in a very fine state of subdivision and hence darker in color than when larger particles are present. When the leaching action takes place, the color changes from gray black to hematite red, probably due to an increase in particle size.

SUMMARY

Several compounds have been synthesized by heating the constituent oxides in intimate contact at temperatures from 650° to 950° C.

Zinc ferrite, artificial Franklinite, $\text{Zn(FeO}_2)_2$, was prepared by heating an intimately mixed powder of zinc oxide and ferric oxide at or above 650° C. The reaction is rapid, being complete at the end of 1 or 2 hr. Continued heating, or higher temperatures of heating, does not affect the formation of the compound. Zinc ferrite is a yellowish brown powder, non magnetic when formed by this method. It is insoluble in alkalis and dilute acids but soluble in concentrated hydrochloric acid.

Magnesium ferrite, $\text{Mg(FeO}_2)_2$, may be prepared in an analogous manner. Its properties are practically identical with those of zinc ferrite.

Calcium ferrite was prepared in the same manner as the above except that calcium hydroxide, instead of the oxide, was heated with ferric oxide. Calcium ferrite is decomposed by boiling in dilute acid. It is reddish yellow and nonmagnetic.

Zinc ferrite may be decomposed by mixing with calcium oxide or magnesium oxide and heating to 850° C. The reaction, however, is not complete at this temperature. The fact that zinc ferrite can be decomposed indicates a possibility of increasing the recovery of zinc by such a method.

X-ray spectrographs were made of the various compounds studied.

ACKNOWLEDGMENTS

The authors wish to thank Prof. G. J. Barker, of the Department of Metallurgical Engineering of the University of Wisconsin, for suggestions given during the investigation.

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DISCUSSION

H. R. HANLEY, C. Y. CLAYTON and D. F. WALSHE, Rolla, Mo. (written discussion). This general subject has been considered of interest and importance by the writers of this discussion for some years, and work has been done leading to a quantitative expression for the influence of iron on the formation of insoluble zinc compounds during roasting. The main part of this work will be published in the future.

In so far as formation of the compound $\text{ZnO} \cdot \text{Fe}_2\text{O}_3$ as an entity, we are in agreement with Swartz and Krauskopf, but the results of our work on the influence of time and temperature on the quantitative formation of ferrite are at variance with these authors.

It is realized that their endeavor was not so much the quantitative expression of ferrite formation with respect to time and temperature as it was to prove the formation

of the compound at certain temperatures, and the decomposition of it by other compounds.

Referring to Table 1, the molecular ratios of ZnO and Fe_2O_3 used are noted as 4, 32, 256 and 400 for samples 1, 2, 3, 4, 5, respectively. These ratios may be calculated from the total ZnO and Fe_2O_3 found by analysis, and when that is considered these molecular ratios are found to be approximately 4, 32, 32.5 and 95, respectively, instead of the four mentioned. However, this does not change the status of the result, which clearly shows the molal ratio of ZnO to Fe_2O_3 in the leached residue to be 1 to 1.

Referring to Table 2, on the subject of the formation of zinc ferrite at various temperatures there is shown that 650°C. is the lowest temperature at which ferrite forms, whereas our results show an appreciable amount formed at 625°C. This table also shows that there is 100 per cent. ferrite formation at 650°C. and 975°C.

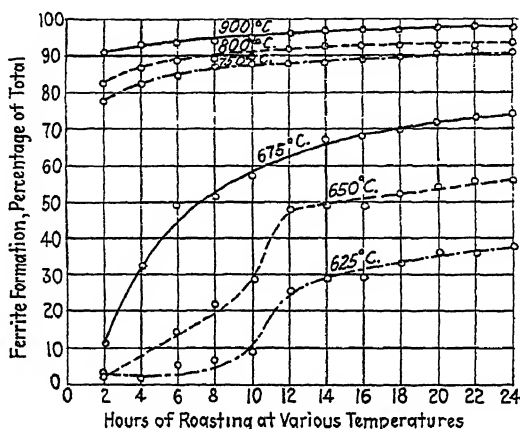


FIG. 8.

for a 2-hr. baking, whereas our results show that there is only about 3 per cent. ferrite formation at 650°C. and about 95 per cent. at 975°C. The detail of this deduction is as follows: For sample No. 4, Table 2 shows 0.0614 g. Fe_2O_3 .

$$0.0614 \frac{81.4}{160} = 0.0312 \text{ g. insoluble Zn theoretically found.}$$

The amount of insoluble Zn actually found was 0.0315, which corresponds with 100 per cent. ferrite formation.

Our numerous results in baking molecular quantities of ZnO and Fe_2O_3 from 2 to 24 hr. at from 625° to 900° clearly show that there is a progressive increase in ferrite with respect to both temperature and time, as may be seen in Figs. 8 and 9, plotted from Table 11.

Table 3, showing the formation of ferrite with various lengths of heating periods at 850°C. , shows 100 per cent. ferrite formation for 2 hr. and over, whereas we obtain 86 per cent. for 2 hr. and about 95 per cent. for 24 hr., as may be seen in Figs. 8 and 9. None of our work extended beyond 24 hr., but the flatness of the curves from 12 to 24 hr. at this temperature would indicate no appreciable rise in ferrite formation beyond 24 hr. Our furnace was heavily insulated, controlled by a Leeds & Northrup potentiometer connected to a control panel.

Figs. 8 and 9 show the quantitative formation of ferrite produced when baking molal quantities of ZnO and Fe_2O_3 from 2 to 24 hr. at various temperatures from 625°C. to 900°C. This formation is expressed as percentage of ferrite formed with

TABLE 10.—*Tests on Series ZnO.Fe₂O₃ Showing Percentage of Ferrite Formed by Baking*
Section A

102 g. ZnO + 206 g. rouge were mixed with water triturated $\frac{1}{2}$ hr., dried to constant weight (no loss at 700° C., 3 hr.). 5-g. samples baked from 2 to 24 hr. at various temperatures. No rabbling. Hump furnace Leads & Northrup potentiometer control.
 Zn in 5-g. sample = 1.330 g.
 Fe in 5-g. sample = 2.285 g.
 Assay total Zn = 26.60 per cent.
 Theoretic insoluble Zn (Fe \times 0.584) = 1.33 g.

Hours Roast.	0	2	4	6	8	10	12	11	16	18	20	22	24
Assay NH ₄ Cl, soluble Zn, per cent.	26.5	26.1	25.9	25.2	24.9	24.1	19.6	18.9	18.8	17.8	17.0	16.9	16.5
Solubility (100 per cent. to insolubility)	99.97	98.15	97.45	94.80	93.6	90.70	73.70	71.10	70.70	66.95	63.95	63.38	62.10
Insolubility (100 per cent. to solubility)	0.03	1.85	2.55	5.20	6.40	9.30	26.30	28.90	29.30	33.05	36.05	36.42	37.90
Insoluble Zn determined, grams.	Trace.	0.0246	0.0339	0.0692	0.0852	0.1236	0.350	0.3845	0.390	0.439	0.479	0.484	0.50
Ferrite formed, percentage of theoretic.		1.85	2.55	5.20	6.40	9.30	26.30	28.90	29.30	33.05	36.05	36.42	37.90

Section B

Same as Section A except temperature which was increased to 650° C. molecular ratios.

Assay NH ₄ Cl, soluble Zn.	26.00	24.50	22.60	20.70	18.80	13.80	13.40	13.40	12.70	12.20	11.60	11.60
Solubility (100 per cent. to insolubility)	97.75	92.15	85.00	77.90	70.75	51.90	50.40	50.40	47.80	45.85	43.65	43.65
Insolubility (100 per cent. to solubility)	2.25	7.85	15.00	22.10	29.25	48.10	49.60	49.60	52.20	54.41	50.35	50.35
Insoluble Zn determined, grams.	0.029	0.104	0.0915	0.292	0.389	0.640	0.660	0.660	0.695	0.724	0.749	0.749
Ferrite formed, percentage of total	2.25	7.85	15.00	22.10	29.25	48.10	49.60	49.60	52.20	54.41	50.35	50.35

Section C

Same as Section A except temperature, which was increased to 675° C.

Assay NH ₄ Cl, soluble Zn.	23.60	17.80	14.50	12.80	11.30	9.40	8.90	8.40	7.90	7.60	7.20	6.90
Solubility (100 per cent. to insolubility)	88.70	66.95	54.55	48.10	42.50	35.30	33.45	31.55	29.70	28.55	27.05	25.92
Ferrite (percentage of theoretic)	11.30	33.05	49.45	51.90	57.50	64.70	66.90	68.45	70.30	71.45	72.95	74.08

Section D

Same as Section A except temperature, which was increased to 750° C.

Assay NH_4Cl , soluble Zn.....	5.95	4.75	4.00	3.55	3.30	3.30	3.05	2.90	2.70	2.50	2.40
Solubility (100 per cent. to insolubility).....	22.35	17.85	15.05	13.35	12.41	12.41	11.47	10.90	10.15	9.41	9.04
Ferrite (percentage of theoretic).....	77.65	82.15	84.95	86.65	87.50	87.50	88.53	89.10	89.85	90.58	90.96

Section E

Same as Section A except temperature, which was increased to 800° C.

Assay NH_4Cl , soluble Zn.....	4.60	3.50	3.15	3.00	2.90	2.30	2.05	2.00	1.95	1.90	1.85
Solubility (100 per cent. to insolubility).....	17.28	13.15	11.85	11.28	10.90	8.05	7.72	7.50	7.31	7.15	6.97
Ferrite (100 per cent. to solubility).....	82.72	86.85	88.15	88.72	89.10	91.35	92.28	92.50	92.66	92.85	93.03

Section F

Same as Section A except temperature, which was increased to 900° C.

Assay NH_4Cl , soluble Zn.....	2.45	1.95	1.65	1.05	1.45	1.10	0.95	0.90	0.85	0.70	0.65
Solubility 100 per cent. to insolubility.....	9.22	7.34	6.21	6.21	5.45	4.13	3.57	3.37	3.19	2.63	2.41
Ferrite, percentage of theoretic.....	90.78	92.06	93.79	93.79	94.55	95.87	96.43	96.63	96.81	97.37	97.56

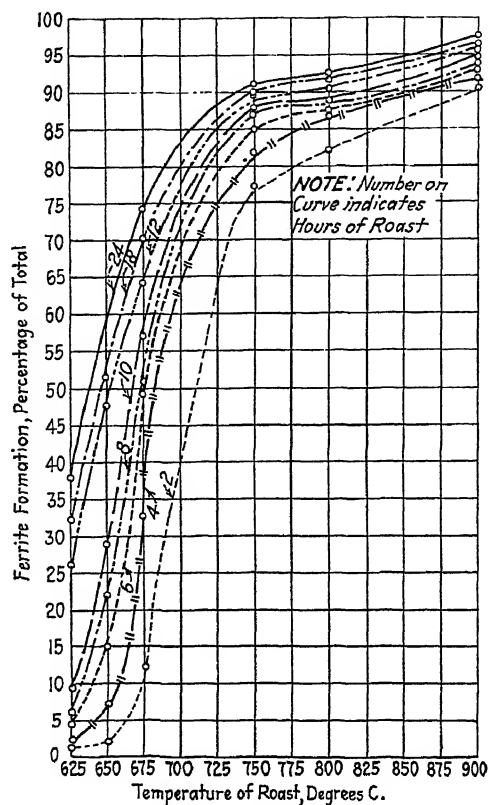


FIG. 9.

TABLE 11.—Rearrangement of Figures for Plotting Ferrite Formation against Temperature^a

Hours Roast	Ferrite Formation, Percentage of Total, at					
	625° C.	650° C.	675° C.	750° C.	800° C.	900° C.
2	1.85	2.25	11.30	77.65	82.72	90.78
4	2.55	7.85	33.05	82.15	86.85	92.66
6	5.20	15.00	49.45	84.95	88.15	93.79
8	6.40	22.10	51.90	86.65	88.72	93.79
10	9.30	29.25	57.50	87.59	89.10	94.55
12	26.30	48.10	64.70	87.59	91.35	95.87
14	28.90	49.60	66.90	88.53	92.28	96.43
16	29.30	49.60	68.45	89.10	92.50	96.63
18	33.05	52.20	70.30	89.85	92.66	96.81
20	36.05	54.41	71.45	90.59	92.85	97.37
22	36.42	56.35	72.95	90.96	92.85	97.56
24	37.90	56.35	74.08	90.96	93.03	97.56

^a Data same as Table 10.

respect to the total possible that could be formed referred to the Fe_2O_3 as a basis (see Tables 10 and 11).

NOMENCLATURE OF THE COMPOUNDS OF IRON ACIDS

There has been some confusion in the literature in regard to the appropriate names for compounds of iron acids. The names "ferrite" and "ferrate" have been used to designate one and the same compound in which the iron functions as the acid radical, whereas these names denote different stages of oxidation of the compound, or, expressed in another way, different valencies of the iron are represented and hence these substances are different compounds. Following are some of the acids, hypothetical or real, to which the salts may be referred:

Hypoferrous acid,	H_2FeO_2 .	Valence of Fe = 2.
Ferrous acid,	$\text{H}_2\text{Fe}_2\text{O}_4$.	Valence of Fe = 3.
Ferric acid,	H_2FeO_4 .	Valence of Fe = 6.

Representing these acids by the following arrangement we have:

Hypoferrous acid,	$\text{H}_2\text{O}.\text{FeO}$
Ferrous acid,	$\text{H}_2\text{O}.\text{Fe}_2\text{O}_3$
Ferric acid,	$\text{H}_2\text{O}.\text{FeO}_3$

If we follow the nomenclature of the chromium compounds in the naming of these iron compounds there would not be any confusion:

CrO	= chromous oxide
Cr_2O_3	= chromic oxide forms chromites
CrO_3	= chromic anhydride forms chromates or trioxide

It would appear plausible therefore to name bivalent iron FeO (acid radical) as *hypoferrite*; the trivalent iron as *ferrite* and the iron with a valence of 6 as *ferrate*.

The zinc compounds accordingly would have the following names:

$\text{ZnO}.\text{FeO}$	= zinc hypoferrite
$\text{ZnO}.\text{Fe}_2\text{O}_3$	= zinc ferrite
$\text{ZnO}.\text{FeO}_3$	= zinc ferrate

The name "ferrite" also appears in the metallographic study of steels and denotes (pure) carbonless iron when it is considered as a microconstituent of steel. The application of the name has no significance to the compound referred to above, but it is deeply rooted in metallographic circles and will remain. It is unfortunate that there are two identical names meaning two entirely different things. The use of a qualifying adjective, preferably the name of the metal combined with the iron, will avoid all confusion; *viz.*, "zinc ferrite."

EXPERIMENTAL DETAIL WORK HEATING MOLECULAR MIXTURES OF $\text{ZnO}.\text{Fe}_2\text{O}_3$ FROM 625° TO 900° C.

In order to obtain uniformity of the mixture of ZnO and Fe_2O_3 , it became necessary to mix a large amount of these reagents with water and grind the paste in a large mortar. This paste was then dried to constant weight of 120° C. A sample was heated to 700° C. for 3 hr. and there was no loss in weight. Five-gram samples were taken for roasts, and placed in scorifiers. Every 2 hr. a scorifier was removed for examination. This procedure gave 12 samples, varying by 2-hr. intervals, from 2 to 24 hr. for a specific temperature. The temperature was controlled by a Leeds & Northrup potentiometer cutting the current in or out as slight variations occurred in the furnace. The furnace used was a Leeds & Northrup hump furnace of standard design. Careful tests showed that no temperature gradient existed in the furnace.

The subject of decomposition of zinc ferrite has been treated in an interesting manner by Swartz and Krauskopf. The writers of this discussion have tested only the influence of hydrogen and SO_2 and the decomposition.

REDUCTION OF FERRITE BY HYDROGEN

The reduction of ferrite by hydrogen was determined by treating a mixture of ferrite and zinc dust with 4.78 per cent. sulfuric acid and comparing the amount of zinc dissolved with that dissolved by a straight 4.78 per cent. sulfuric acid leach of the same ferrite. The ferrite was made by baking a mixture of zinc oxide and ferric oxide at 900°C . for 6 hr. Total zinc in this mixture was 25.9 per cent. Zinc soluble in ammonium chloride was 1.84 per cent. and insoluble zinc was 24.06 per cent.

In tests 1 and 2 (duplicate) 0.500 g. of ferrite was leached without the addition of zinc dust in 175 c.c. of 4.78 per cent. sulfuric acid at a temperature of 55°C . for a period of 50 min. with the following results:

TEST NUMBER	SOLUBLE IRON, GRAMS	SOLUBLE ZINC, GRAMS	RESIDUAL ZINC, GRAMS
1	0.03899	0.0313	0.0984
2	0.0390	0.314	0.0985

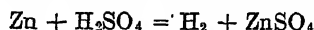
In tests 3 and 4 (duplicate) 0.550 g. of zinc dust was used in addition to the components of the previous tests, with the following results:

TEST NUMBER	SOLUBLE IRON, GRAMS	SOLUBLE ZINC, GRAMS	RESIDUAL ZINC, GRAMS
3	0.0631	0.5989	0.0810
4	0.0633	0.5987	0.0808

The following is a summary of the four tests:

TEST NUMBER	SOLUBLE IRON, GRAMS	SOLUBLE ZINC, GRAMS	RESIDUAL ZINC, GRAMS	TOTAL ZINC, GRAMS
1 and 2	0.0389	0.0313	0.0984	0.1297
3 and 4	0.0632	0.5988	0.0809	0.6797

The amount of zinc insoluble in the residue from the leaching of ferrite in the presence of zinc dust is much less than when no zinc dust is added. This shows that ferrite is reduced by the hydrogen resulting from the reaction:



A further confirmation of this is gained by inspection of the relative amounts of iron leached in the two cases. There was an excess of 0.0243 g. iron leached in the sample containing the zinc dust; viz., $0.0632 - 0.0389 = 0.0243$. The zinc previously combined with this iron in the form of $\text{ZnO} \cdot \text{Fe}_2\text{O}_3$, but now made soluble was

$0.0243 \times 0.584 =$	0.01434
The original H_2SO_4 soluble zinc	0.03130
Total zinc soluble in tests 3 and 4	0.04564

The zinc made soluble by hydrogen attack was

$$\frac{0.01434}{0.0984} = 14.34 \text{ per cent. of the insoluble zinc.}$$

Final summary is:

TEST NUMBER	SOLUBLE ZINC, GRAMS	INSOLUBLE ZINC, GRAMS	TOTAL ZINC, GRAMS
1 and 2	0.0315	0.0984	0.1297
3 and 4	0.0456	0.0809	0.6797

CONCLUSIONS

It is evident from these tests that hydrogen gas generated in the acid-leach suspension due to the reaction between zinc dust and sulfuric acid reduces ferrite and causes this portion of the zinc to go into solution.

Reduction Effect of SO₂

It is generally supposed that SO₂ gas will reduce the amount of ferrite formed in the calcine. The effect of this gas at a concentration of 6 per cent. by volume introduced over a bed of ferrite maintained at 650° C., and rabbled at intervals of 10 min., is practically nil. The SO₂ was supplied from a bomb of liquid SO₂. The above conclusion was made from a test in which the above concentration of SO₂ was maintained under the above conditions for 6 hours.

Inasmuch as it is common knowledge that the interstitial SO₂ gas within the bed of sulfide ore during the process of roasting reaches concentrations two or three times that of the atmosphere above the bed, it was considered that this high concentration (13 to 14 per cent. SO₂) was necessary to affect a reduction of the ferrite. Accordingly a 2-g. sample of pure ferrite was mixed with 4 g. pyrite and roasted for 8 hr. at 650° C. The calcine produced in this test was leached with Low's solution, the filtrate from which contained zinc to the extent of 44.3 per cent. of the total zinc present in the original ferrite. While no extensive work was done on this part of the subject, it seems certain that direct contact with the rich interstitial SO₂ gas is necessary for decomposition when this reagent is used.

O. C. RALSTON, Berkeley, Calif. (written discussion).—The author's X-ray data tending to confirm the existence of zinc and magnesium ferrates as definite chemical compounds are welcome. Of even greater interest to me has been their discovery that in formation of zinc ferrate from zinc oxide and ferric oxide, no reaction took place below about 650° C. This observation is important when considered in connection with certain physical properties of ferric oxide. I have recently been engaged in preparing a critical report on ferric oxide and wish to present in abstract a few properties of interest in the above connection. (My forthcoming paper will give the references.⁴)

Ferric oxide suffers a transition at 678° C. which absorbs about 6690 cal. per mol, causes a marked change in direction of a differential heating curve, corresponds to a rapid drop in magnetic susceptibility, causes a different thermal coefficient of expansion above that temperature, and the thermal conductivity of the high temperature form of ferric oxide increases with temperature at a more rapid rate than that of the low-temperature form. No X-ray crystallographic work has been done on the two forms of ferric oxide above and below 678° C., but the other physical changes are sufficiently profound to make it probable that the 678° transition is a polymorphic one.

The temperature of this transition is lowered by impurities: magnetite up to about 10 mol per cent. lowers the transition temperature till 650° C. is reached, beyond which further magnetite will not dissolve and so cannot further lower the transition temperature. Alumina is taken up by ferric oxide until it is saturated with 12 molecular per cent. Al₂O₃ and the transition temperature of the Fe₂O₃ has been lowered to 575°. Chromic oxide functions in a similar fashion but is more soluble, the temperature of the transition being lowered to 230° C. by 40 mol per cent. of Cr₂O₃. On account of these effects of impurities it is not surprising to find the transition temperature as observed by different experimenters, using different physical properties, reported all the way from 640° to over 700° C. The most precise work,

⁴O. C. Ralston: Iron Oxide Reduction Equilibria. U. S. Bur. Mines *Bulletin* in preparation.

that of Sosman in the Geophysical Laboratory, places the transition temperature of pure ferric oxide at 678°C .

The difference in reactivity of ferric oxide toward zinc oxide above and below 650°C ., as reported by the authors, suggests at once some connection with the 678°C . transition temperature. The precautions in the use of their chromel-alumel thermocouples are not mentioned by the authors nor are any corrections mentioned. If the couples were merely stuck into the muffle in a straight line, the thermal conductivity of the wires would cool the junction below the temperature of the muffle and this would be especially true in case the thermocouples were protected by a pyrometer tube. The probability is, therefore, that the temperature 650°C ., as reported by the authors, is somewhat below the actual temperature of the muffle and correction of the approximate readings would probably give a value much nearer 678° .

Without attempting to theorize too much on scanty data, it is well to point out that this is one of the first bits of (apparent) chemical evidence to parallel the differences in physical properties of the two forms of ferric oxide above and below 678°C . I suggest that possibly the chemical reactivity is developed only when ferric oxide is in a disorganized state at 678°C ., passing from one crystalline form to the other, and if so the proper heat treatment would be one of repeatedly raising and lowering the temperature through 678°C . if a maximum ferrite production were demanded and conversely for minimum ferrite formation to either refrain from raising the temperature to this point or to go to higher temperatures very rapidly. In this connection it is well to point out other apparent thermal points of ferric oxide, one at 360°C . and the other at about 940°C .

J. H. HASTINGS, La Porte, Colo. (written discussion).—This interesting paper prompts the suggestion of the part that zinc ferrite plays in the retorting of zinc ores. Although the following remarks have not been confirmed by experimental data, the results have been noted in actual practice.

When zinc concentrates of the Tri-State district were roasted in a Hegeler kiln, if the roast was carried only to a point where the sulfur in a sulfide condition was at least 1 per cent., the reducing action of the sulfide sulfur was sufficient to prevent the formation of zinc ferrite to any extent, while if the roast was continued to a point where practically all the sulfide sulfur was eliminated, zinc ferrite formed.

Probably the best condition for the prevention of the formation of zinc ferrite would be in a roaster in which the last per cent. of sulfide sulfur would be changed over to zinc sulfate, and then the zinc sulfate would be broken up by heat without excess of oxygen. This practice has been followed in cases where carbon in the form of coal or coke was introduced on the bottom hearth. In this last method probably there is likely to be a loss of zinc due to the reduction of some zinc oxide. At least there should be sufficient air to roast the zinc sulfide and in the last stages to leave the zinc mostly in the form of sulfate to be finished off without an excessive amount of air.

Should the roast be carried sufficiently far under conditions favorable for the formation of zinc ferrite, the roasted ore comes from the roaster with a coating containing zinc ferrite on each grain. The presence of the zinc ferrite has a retarding influence on the reduction of zinc. There will thus be a slowing down of the reducing process in the initial stages at least.

In some cases the roasting is carried on too hot and there may be a fused coating on the ore particles but in that case the iron would be in a ferrous state for favorable conditions for the formation of a ferrous silicate.

In treating zinc concentrates in a Dwight-Lloyd sintering machine, there is probably no zinc ferrite formed and the iron will be in the form of magnetite to a large extent. This sinter always works easily in the retorts, because of its porous nature and also on account of the absence of zinc ferrite.

It is possible that if zinc ferrite is heated with ferrous oxide or ferrous sulfate in the retort, the zinc ferrite will break up with the formation of magnetite and zinc oxide, which then will be further reduced by the carbon.

It is possible also that the addition of magnesium oxide to the retort charge might improve the ore after certain conditions in roasting, for the better recovery of zinc.

M. F. COOLBAUGH, Golden, Colo.—The subject of ferrites has been a very interesting one to me, and I feel that there are many factors in the formation of these compounds that have not been clearly discussed. Undoubtedly there are many other factors which are entirely unknown, and the whole subject seems worthy of careful study.

The effect of impurities in the formation of ferrites is a vital one. Zinc ferrites formed under constant conditions of temperature and time from strictly pure iron oxides and zinc oxides are not the same as those formed from the same materials when small impurities of alumina or silica, or both, are present. They vary in solubilities, in magnetic properties, and in the readiness with which they can be decomposed.

I agree with Mr. Hanley that SO_2 (but more likely SO_3) is responsible for the decomposition of ferrites. There are, however, certain ferrites which are decomposed with a very low percentage of SO_2 (or the accompanying SO_3) and there are other ferrites which will not decompose at all under like conditions.

The following are some of the general results of experiments obtained in our laboratories some years ago: A strictly pure ferric oxide formed by roasting ferrous oxalate was mixed with chemically pure zinc oxide and ignited. This produced a ferrite not very magnetic but very difficult to decompose by sulfur dioxide or sulfur trioxide, even though treated for long periods with these reagents. The product formed from the same iron oxide and zinc oxide at the same temperature in the presence of small quantities of aluminum oxide or silica or both was decidedly different. The total amount of the aluminum and silica employed was small, less than 2 per cent. The ferrite under the latter condition was much more soluble in acid and was decomposed readily by sulfur dioxide or sulfur trioxide. The differences in magnetic properties were not carefully studied.

The paper of Swartz and Krauskopf says that zinc ferrites are not magnetic under the conditions employed in their formation. This is undoubtedly true, but there are many conditions of formation under which magnetic ferrites will form.

I am rather surprised to find no references made to the formation of copper ferrites, if that is the proper name to use. We found that the insoluble compounds of copper and iron form readily and that they decompose more readily than zinc ferrites.

I believe that an extensive study of these insoluble compounds would prove very valuable. If the formation of ferrites can be prevented or methods can be derived for decomposing them after they have once been formed, a real contribution to hydrometallurgy will be made.

G. L. OLDRIGHT, Salt Lake City, Utah.—I have a number of irons in the fire along this same line, but I do not want to say too much because some of the experimental work is not yet completed. However, there are some things in this paper that should be checked up. In the first place, about the proportion that is combined, without saying what the compound is, if iron and zinc oxides made from calcining pyrite and sphalerite are heated together in varying proportions and at varying meshes and at various temperatures, it will be found, as would be suspected with a solid-solid reaction, that the amount of insoluble compound will be a function of the surface exposed and of the time the constituents are exposed, of their intimacy of contact and of the temperature employed—these four variables.

Also, again without saying anything as to the compound, more than one zinc can be tied up with the iron; that is, more than one ZnO with one Fe_2O_3 . We are interested in the matter of the solubility of the zinc compounds in roasting in dilute acid. On going from hearth to hearth and taking samples from a large roaster, and then analyzing them, using Low's reagent of ammonia plus ammonium chloride to give the free zinc, and sulfuric acid, which gives the free zinc plus a certain amount of what we call ferrites, it will be found that on the top hearth the difference in the solubility of the zinc given by these two solvents is quite large. On going down towards the bottom hearth, the difference between the amounts of zinc soluble in the two reagents becomes less and less; in other words, as the ferrites get baked more and more, they become more and more insoluble.

Just a word as to the relative merits of lime and magnesia: In so far as leaching would be concerned, if a very great deal of sulfuric acid is made in what is usually called the Coolbaugh process (that of Coolbaugh and Read and their coworkers) one should have enough acid to take care of the lime. Should magnesia be added instead of lime to make the ferrite soluble, however, I think Professor Coolbaugh will bear me out in saying that magnesia in solution is "not so good."

W. SEGUINE, JR., Wilmington, Del.—Lithopone manufacturers will agree with me that Joplin concentrates furnish one of the best zinc materials for lithopone, but this material is expensive in the East on account of freight and roasting charges. Sulfate roasting saves sulfuric acid, but with Joplin concentrates the absence of iron inhibits satisfactory sulfation. I suggest that coarse pyrite be added to the roaster feed, in order to obtain this necessary iron for sulfation in a furnace of the down-draft type such as has been developed by Mr. Coolbaugh and his coworkers. I would like to see some experimental work done along these lines.

C. P. LINVILLE, Bound Brook, N. J.—On the curve that Dr. Hanley showed, it seems to me that a question of chemistry ought to be brought up, as to why the reaction, which hardly proceeded at all during 12 hr. of heating, should suddenly take place in the thirteenth hour. It seems rather strange for materials to lie for 12 hr. without a chemical reaction and then suddenly to react very rapidly. The peculiar shape of the curve seems to call for something in the way of an explanation of the chemistry of that change.

H. R. HANLEY, Rolla, Mo.—In regard to the formation of sulfates in the Joplin ores, it is generally conceded that there would be very little sulfate formed because of a lack of iron, but actually under definite tests I have proved that this is not so.

The Joplin concentrate containing only about 0.6 per cent. iron can be roasted at 750°C . in a circular-hearth furnace and obtain a calcine that will contain 4 per cent. zinc as zinc sulfate and 0.3 to 0.4 per cent. zinc as sulfide. This result has been accomplished daily over a period of several days. The roaster referred to is an electric muffle containing a thermalloy heating element below a carborundum hearth on which the ore is placed. The rabbling of the ore during roasting is accomplished by specially designed plows connected to a revolving arm. These plows scrape the ore next to the bottom of the hearth and bring it to the top. Analyses of the gas from the roasting operation was made almost continuously. This furnace is similar to the one designed by C. A. Hansen,⁵ although much larger, and will roast 30 lb. ore at one time.

Referring to the early electrolytic zinc process operations, the roasting was performed with a restricted amount of air. This procedure caused relatively high temperatures and low sulfate, also considerable insoluble zinc because of the impurities present, chiefly iron. Later the roasting department commenced using a sulfating

⁵ C. A. Hansen: *Electrolytic Zinc. Trans. (1920) 64, 85.*

roast. This was brought about by increasing the air, which reduced the temperature, particularly the second hearth where so many exothermic reactions are going on. If this hearth is cooled to a reasonable temperature, for example 650° to 700° C., a high degree of sulfation will result, but it has been found that by this method too much sulfate is formed. The roasting, then, went from one extreme to another. Fortunately selective flotation had begun to produce cleaner concentrate, *i. e.*, lower in iron, and this type of concentrate permitted the raising of the temperature to decompose the excess sulfate without inordinately forming ferrites. There may be considerable variation in the temperature of the roast, provided the concentrate is clean.

Reduction of Roasted Cassiterite Concentrates

By W. W. LOO* AND E. F. KERN,† NEW YORK, N. Y.

(New York Meeting, February, 1928)

A REVIEW of the literature on the reduction of cassiterite showed that scarcely any progressive changes were made in the methods of reducing cassiterite until within the last two decades, and that during that period many improvements in the metallurgy of tin have taken place. Since 1914 the world's consumption of tin has increased considerably, resulting in the exploitation of primary tin deposits such as the Bolivian lode, which has become an important source of tin. On account of the low tin content of Bolivian ore, and the presence of objectionable impurities of the reduced tin, the smelting of this ore and the refining of the tin has changed considerably from the methods formerly used.

A comparative study of the methods of smelting cassiterite concentrates suggested that an investigation be planned so as to determine what factors effect the extraction of tin from its ores by reduction with carbon. The first part of the experimental work was the determination of whether roasted cassiterite concentrates can be reduced by carbon without the use of flux, and the reduced pellets of tin be collected into a single mass separate from the dry gangue residue. The second part of the experimental work was a comparison of the effect of variations of the different factors on the smelting of roasted cassiterite concentrates: first, the effect of varying the composition of the slag; second, the effect of varying the amount of reducing agent; third, the effect of varying the relative amount of the resulting slag; and fourth, the effect of different kinds of reducing agents.

REVIEW OF LITERATURE

Experiments on the smelting of Bolivian tin concentrates were made by James P. Gill in 1920 in the metallurgical laboratory of the School of Mines, and submitted as a graduate thesis. The results of these tests are fully reported as they are preliminary to the experimental part of this paper, and also because they have not been published. The variables investigated by Mr. Gill were: (a) the effect of temperature on the rate of reduction of cassiterite concentrates; (b) the time required for

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complete reduction; (c) the effect of the relative amount of slag upon the tin extraction; and (d) the effect of variations of the fluxes. The cassiterite concentrate contained 56.8 per cent. tin, 18.0 per cent. SiO_2 , and 4.7 per cent. iron in the form of ferric oxide, equivalent to 6.0 per cent.

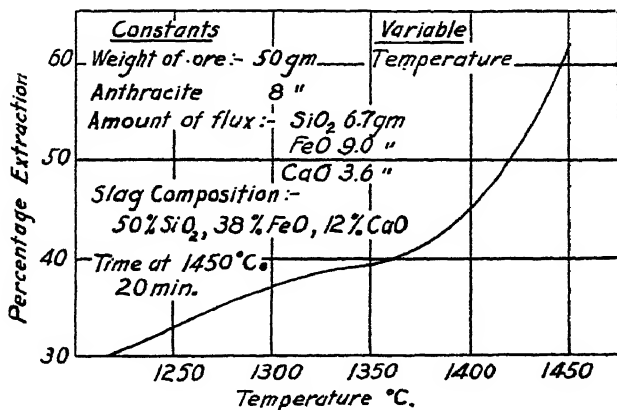


FIG. 1.—EFFECT OF TEMPERATURE ON THE REDUCTION OF CASSITERITE CONCENTRATES.

FeO . Anthracite was used for the reducing agent. The tests were made in crucibles, using 50 g. of the cassiterite concentrates as the unit charge. The results are shown by the curves of Figs. 1, 2, 3 and 4, by the figures in Table 1, and the following data:

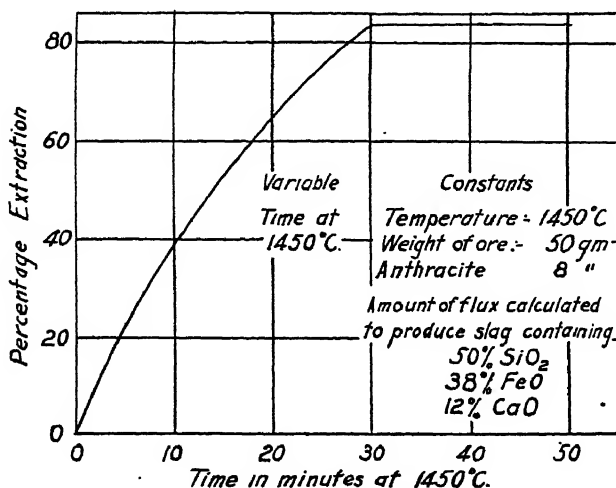


FIG. 2.—EFFECT OF TIME ON THE REDUCTION OF CASSITERITE CONCENTRATES.

(a) The curve (Fig. 1) shows that the percentage extraction of tin from the concentrates increased with increase of temperature from 1200° C. to 1450° C., when the charges were made to produce a slag of ratio composition 50 per cent. SiO_2 , 38 per cent. FeO , and 12 per cent. CaO . This

slag was selected because slags of this composition were produced at a smelter treating Bolivian tin concentrates.

(b) Although the complete reduction of cassiterite concentrates by anthracite is dependent on temperature, time is another factor for the complete reduction, as shown by the curve (Fig. 2). About 30 min. was required for the maximum reduction at 1450°C ., when the charges were made to produce a slag containing about 50 per cent. SiO_2 , 38 per cent. FeO , and 12 per cent. CaO .

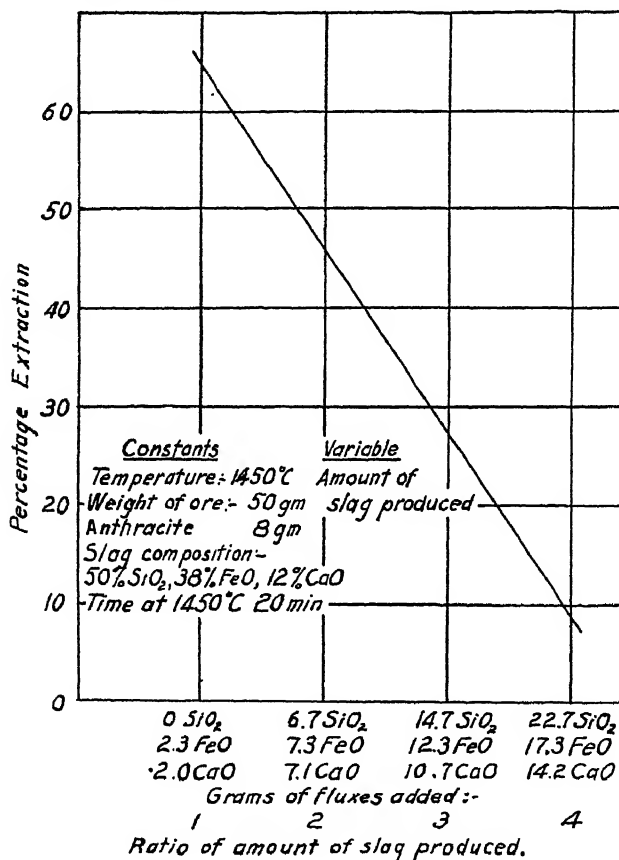


FIG. 3.—EFFECT OF RELATIVE AMOUNT OF THE RESULTING SLAG ON THE EXTRACTION OF TIN FROM CASSITERITE CONCENTRATES.

(c) The curve (Fig. 3) shows that the amount of tin extracted decreases rapidly with increase in amount of the resulting slag.

(d) In the smelting of an ore, the first consideration is the kind and amount of fluxes required in order to obtain the highest extraction of the metal. The fluxes used were silica, iron oxide, lime, and magnesia in varying amounts as given in Table 1. The charges were brought to 1450°C . in about 45 min. and held at this temperature for 20 min. The

results are given in Table 1, and shown by the curves of Fig. 4. An excess of either silica or iron oxide caused a large loss of tin; whereas an increase of lime or magnesia flux gave higher extractions of tin. High tin extractions resulted when the slags contained between 33 and 41 per cent. silica, 11 and 25 per cent. ferrous oxide, and the remainder lime and magnesia.

United States Patent No. 1544198 of June 30, 1925, was granted to Edwin B. Thornhill for a process of reduction of tin ores by solid fuel

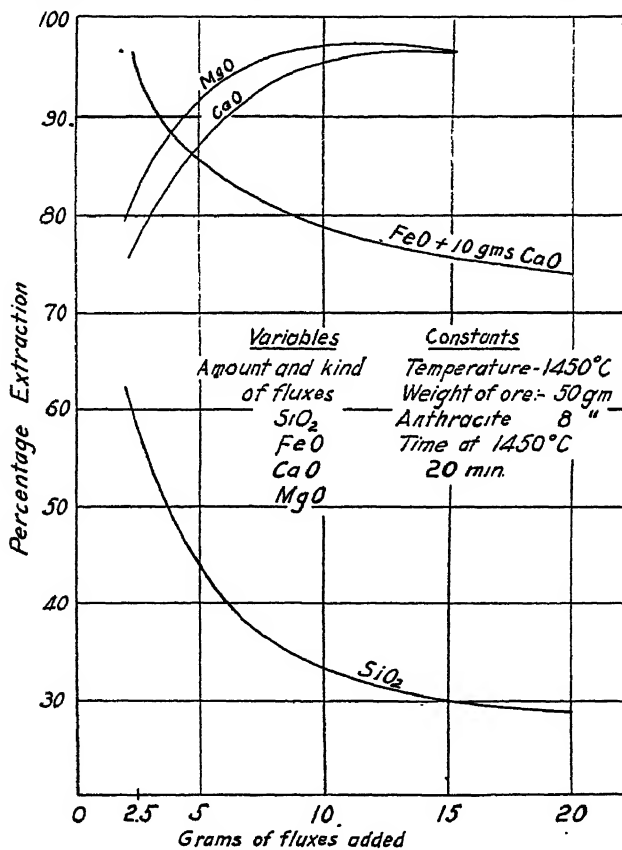


FIG. 4.—EFFECT OF THE KIND AND AMOUNT OF FLUX ON THE EXTRACTION OF TIN FROM CASSITERITE CONCENTRATES.

without the production of slags. The tin ore, mixed with solid fuel, is heated in a reducing atmosphere in a suitable furnace with a revolving disk hearth provided with a pervious bed for supporting the charge, and suitable rabbles for distributing the charge. The furnace bed is made of a layer of fragments of an alloy of iron and tin ("hard-head") of sufficient porosity to permit the melted tin to percolate through it. The reducing operation is effective at a temperature of about 1000° C. liberating about

TABLE 1.—*Effect of Variation of Flux on the Tin Extraction*

Constants: Amount of Anthracite; Temperature 1450° C.; Time 20 Minutes

Relative Amounts of Slagging Constituents in the Charge											Percentage of Tin Extracted
Flux Added	SiO ₂		FeO Equivalent			CaO		MgO			
	Ore Contained, Grams	Amount Added, Grams	Total Per Cent.	Ore Contained, Grams	Amount Added, Grams	Total Per Cent.	Amount Added, Grams	Total Per Cent.	Amount Added, Grams	Total Per Cent.	
SiO ₂	9	2.5	79	3		21					60
	9	5	82	3		18					43
	9	10	86	3		14					33
	9	15	89	3		11					30
	9	20	90	3		10					29
Fe ₂ O ₃ and CaO	9		37	3	2.5	22	10	41			95
	9		33	3	5	30	10	37			86
	9		28	3	10	41	10	31			79
	9		24	3	15	49	10	27			75
	9		21	3	20	55	10	24			74
CaO	9		62	3		21	3	17			80
	9		53	3		18	5	29			87
	9		41	3		14	10	45			96
	9		33	3		11	15	56			97
MgO	9		63	3		23			2.3	14	80
	9		53	3		18			5	29	92
	9		41	3		14			10	45	97.5
	9		33	3		11			15	56	97

80 per cent. of the metallic tin from the ore. The reduced molten tin percolates through the pervious bed to an outlet which delivers it by means of a chute to a suitable receiver from which it is cast into pigs. The remainder of the tin, about 20 per cent., which is in the scoria as grains, pellets or slugs mixed with residual carbon and gangue (on the hearth), is eventually discharged by means of the rabbles. The metallic tin remaining in the scoria is separated by screening, followed by crushing and the usual methods of gravity concentration.

H. H. Alexander and J. R. Stack published a paper¹ in 1924, which is a valuable contribution to modern methods of smelting tin ores and refining of tin. The article describes in detail the smelting and refining plant of the American Smelting & Refining Co., at Perth Amboy, N. J., which was shut down several years ago. Smelting was conducted in reverberatory and in blast furnaces, as follows:

Reverberatory Smelting.—The roasted mixture of concentrates and by-products, containing about 60 per cent. tin and under 0.5 per cent. sulfur, were thoroughly mixed with fine anthracite and the necessary fluxes to give a desired slag. The resulting slag contained about 35 per cent. SiO_2 , 12 per cent. Fe (= 15.5 per cent. FeO), 25 per cent. CaO , and 1.5 per cent. Sn . When smelting concentrates very low in iron, complete reduction of the tin was made in one operation, and the slags which contained up to 2 per cent. tin were discarded. Ores containing 3 per cent. or more of iron required a two-stage smelting operation in order to completely separate iron from the tin: namely, "ore smelting" and "slag smelting." The iron-tin alloy ("hand-head") produced by "slag smelting" was treated separately for the recovery of tin by smelting it with the required flux and tin ore (SnO_2) which oxidized the iron and liberated tin. A method for cleaning tin slags by resmelting with lead oxide to produce solder was tried, and found to give good extraction of tin and lead.

Blast-furnace Smelting.—A mixture of unroasted and roasted concentrates and roasted by-products were agglomerated on a Dwight-Lloyd sintering machine, to render the mixture suitable for blast-furnace charges. Smelting was done in two stages, "ore smelting" and "slag smelting:" the sinter, flux, iron-tin alloy, return slag and coke, were spread and mixed in the proper proportions in a charge car, and fed into the blast furnace. The slag produced was a bisilicate containing approximately 37 per cent. SiO_2 , 20 per cent. FeO , 23 per cent. CaO , 12 per cent. Al_2O_3 , 3 per cent. MgO , and 5 to 7 per cent. Sn . The molten slag from the blast furnace was delivered to a slag-settling reverberatory furnace, and small-sized anthracite, together with crushed lime rock, was added and rapidly stirred into the molten slag. The lime replaced stannous oxide from its combination with silica, and the SnO and some FeO were

¹ H. H. Alexander and J. R. Stack: Reduction and Refining of Tin in the United States. *Trans.* (1924) 70, 404.

simultaneously reduced by the anthracite, forming an iron-tin alloy ("hard-head"). This treatment removed the major part of the tin from the slags (U. S. Patent No. 1487135, Mar. 18, 1924).

Partial Reduction.—A process of partial reduction of the ore was tried, by which the greater portion of the tin was allowed to enter the slag; the metal produced by resmelting this slag, being comparatively free from lead, copper, arsenic, antimony and bismuth, was suitable for "market grade" tin. The metal obtained by the first partial reduction, being relatively impure, was subjected to electrolytic refining to separate the lead, copper, arsenic, antimony, bismuth, and silver.²

W. W. Loo was granted patents³ for the treatment of low-grade oxidized, complex lead-tin ores by sulfatizing the oxidized lead and copper minerals by means of soluble alkali sulfides; separating the major part of the sulfides by flotation, and leaching the tailings with hot, acidified salt solution for the removal of the greater part of remaining lead and copper, leaving a concentrate high in tin and very low in copper and lead, which is suitable for smelting. An oxidized lead-tin ore from Kotchiu, Yunnan, China was treated by this process: the ore contained 23.1 per cent. Sn, 44.6 per cent. Pb, and about 6 oz. Ag per ton; the flotation concentrates contained 3.5 per cent. Sn, 67.2 per cent. Pb, and 7.2 oz. Ag per ton; the flotation tailings contained 44.0 per cent. Sn, 16 per cent. Pb, 9.7 per cent. Fe, and 4.8 oz. Ag per ton; the residue from leaching the flotation tailings with hot, acidified brine solution contained 55.4 per cent. Sn, 0.4 per cent. Pb, 14.0 per cent. Fe, 2.1 per cent. SiO₂, 1.2 per cent. CaO, and a trace of silver. Residues of this composition were smelted, so as to determine whether low-grade lead-tin ores have a possibility of becoming an economical source of tin. The results of these smelting tests are reported later (see Table 9). In some districts of East Asia there are many low-grade, complex lead-tin ores containing from 7 to 45 per cent. Pb, 1 to 5 per cent. Cu, 3 to 4 per cent. Fe, small amounts of silver, and up to 25 per cent. Sn, which have a possibility of being treated by this process.

REDUCTION OF ROASTED CASSITERITE CONCENTRATES WITHOUT FLUX

Experiments were made to determine whether flux was necessary for the reduction of roasted tin concentrates, the idea being that metallic tin can be reduced from its ore by means of suitable reducing agents; no flux being necessary, provided the temperature is kept below the sintering point of the residue.

² H. H. Alexander and J. R. Stack: *Op. cit.*, 432; J. R. Stack: *Electrolytic Refining of Tin*. *Trans. Am. Elec. Chem. Soc.* (1924) 45, 441.

³ U. S. Patent Application Series No. 20625 (1925); Chinese Patent No. 224 (June 19, 1926).

TEMPERATURE FOR COMPLETE REDUCTION

A series of tests was made to determine the temperature at which tin would begin to reduce from its concentrates, and also the time necessary for the complete reduction. The reducing agents tried were anthracite, bituminous coal and charcoal. The charges of roasted cassiterite concentrates and pulverized reducing agent, in proportion to produce metallic tin and CO, were thoroughly mixed and heated in an electric tube furnace for one hour. The results are given in Table 2.

The roasted cassiterite concentrates had the following composition:

	ANALYSIS, PER CENT.		RECAST ANALYSIS, PER CENT.
Sn.....	68.72	SnO ₂	87.50
Fe.....	2.90	FeS.....	0.11
S.....	0.04	Fe ₂ O ₃	4.05 = 3.64 FeO
CaO.....	0.35	CaO.....	0.35
SiO ₂	1.90	SiO ₂	1.90
Undetermined.....	26.90	Undetermined....	6.09
Total.....	100.00		100.00

TABLE 2.—*Temperature of Reduction of Roasted Cassiterite Concentrates*
20 Grams Concentrates Used in Each Test. Charges Not Disturbed during Heating

Test No.	Reducing Agent	Temp. °C.	Reducing Effect
1-a	Anthracite	766	} No apparent reduction
b	Bituminous coal	766	
c	Charcoal	766	
2-a	Anthracite	817	} Very minute Sn pellets formed
b	Bituminous	817	
c	Charcoal	817	
3-a	Anthracite	865	Small tin globules formed
b	Bituminous	865	Small tin globules formed
c	Charcoal	865	Very minute Sn grains formed
4-a	Anthracite	917	} Reduction of Sn complete; collected as globules
b	Bituminous coal	917	
c	Charcoal	917	
5-a	Anthracite	934	} Sn completely reduced, collected as globules
b	Bituminous coal	934	
c	Charcoal	934	
6-a	Anthracite	944	} Sn completely reduced, collected as globules
b	Bituminous coal	944	
c	Charcoal	944	
7-a	Anthracite	985	} Sn completely reduced, collected as globules
b	Bituminous coal	985	
c	Charcoal	985	
8-a	Anthracite	1071	} Temp. too high, reduced Sn and residue fused together
b	Bituminous coal	1071	
c	Charcoal	1071	

These results show that the rapid reduction of tin concentrates by solid carbonaceous reducing agents commenced at about 800° C., and that above 1050° C. the gangue constituent sintered. Therefore if tin ore is reduced by means of pulverized solid, carbonaceous fuel, without any flux, the temperature should be kept between 850° and 950° C. in order to obtain complete reduction of tin and prevent sintering of the gangue residue.

When reduction was carried out at temperatures between 800° and 950° C., the reduced metal formed as very minute pellets and small globules, most of which collected on the side and bottom of the crucible; the higher the temperature, the larger the size of individual particles of reduced tin.

EFFECT OF STIRRING

The pellets of reduced tin were collected into a single mass, by reheating the previous reduced charges to between 500° and 600° C., then stirring the heated porous mass with a thin graphite rod for a short time. With this treatment, the pellets of metal collected as a single mass and the gangue floated as unfused residue on the surface of the molten tin.

The possibility of collecting the numerous isolated tin pellets by reheating and stirring the pre-reduced charges suggested a method for direct reduction of tin from its ore. This method of treatment is designated as "Direct Reduction" process.

The final "direct reduction" tests were made in a manner similar to that outlined above, except that the temperature was kept at about 900° C., and the charge was stirred every five minutes. The charge of roasted tin concentrates was mixed with the required amount of finely divided anthracite to form CO by complete reduction of SnO_2 . Records of the results are given in Table 3.

TABLE 3.—*Reduction of Cassiterite Concentrates without Flux*

Tin Concentrate Used = 100 Grams

Test No.	Anthracite, Per Cent.	Temperature, °C.	Liquated Sn, Grams	Percentage Extraction
1	10	870-900	60.8	88.5
2	10	900	63.0	91.6
3	20	900-910	63.6	92.4

Stirring aided in increasing the reducing effect of the anthracite, as well as collecting the reduced tin pellets into a single mass which finally occupied the lower part of the crucible. The dry, lighter gangue residue floated on top of the molten tin.

Direct-fired rotary kilns, which were found by the Bureau of Mines to be most satisfactory for the reduction of iron ores,⁴ should prove efficient for the "direct reduction" of cassiterite.

SMELTING OF ROASTED CASSITERITE CONCENTRATES

Tests were made to determine the effect of variations of the different factors of the smelting operation on the extraction of tin from roasted cassiterite concentrates: (1) the effect of varying the composition of the slags; (2) the effect of varying the amount of reducing agent; (3) the effect of varying the relative amounts of the resulting slags; (4) the effect of different kinds of reducing agents.

DETERMINATION OF FUSION TEMPERATURES OF FERROUS OXIDE AND LIME SILICATES

The fusion points of a series of ferrous oxide + lime + silica mixtures were determined. The materials used for making the cones for these

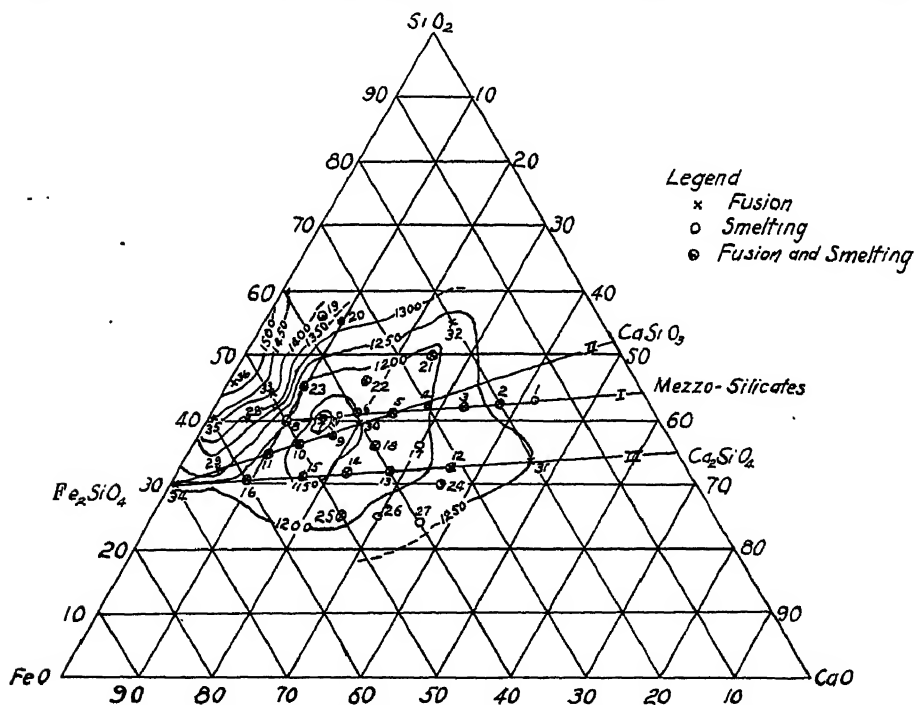


FIG. 5.—FUSION TEMPERATURES OF MIXTURES OF SILICA, FERROUS OXIDE AND LIME.

determinations were pulverized fused quartz (99.9 + per cent. SiO_2), lime (98 + per cent. CaO), and chemically pure ferrous oxide produced from ferrous oxalate by Baker and Adamson. The materials were ground to pass 100 mesh, and dried. Calculated amounts of the quartz,

⁴ C. E. Williams, E. P. Barrett and B. M. Larsen: Production of Sponge Iron. U. S. Bur. Mines Bull. 270 (1927) 114-150.

TABLE 4.—*Fusion Temperatures of Ferrous Oxide and Lime Silicates*

Group	Mixture No.	Ratio Composition			Fusion Point, ° C.
		SiO ₂	FeO	CaO	
I	2	43	20	37	1270
	3	42	25	33	1220
	5	41	35	24	1180
	6	41	40	19	1155
	7	40	45	15	1120
	8	40	50	10	1200
	28	40	55	5	1320
II	30	39	40	21	1150
	9	38	45	17	1130
	10	36	50	14	1135
	11	35	55	10	1180
	29	32	62	5	1260
III	31	34	20	46	1250
	12	32	32	36	1210
	13	32	40	28	1175
	14	32	46	22	1170
	15	31	52	17	1140
	16	31	60	9	1180
Intermediate	18	36	40	24	1170
IV	32	55	20	25	1235
	20	55	35	10	1310
	21	50	25	25	1185
	22	46	36	18	1175
	23	45	45	10	1195
	33	44	50	6	1400
V	25	25	50	25	1185
	27	24	40	36	1245
Fe ₂ SiO ₄	34	30	70	..	1185
3FeO + 2SiO ₂	35	40	60	..	1425
FeO + SiO ₂	36	45.5	54.5	..	1525

lime, and ferrous oxide were thoroughly mixed, moistened with water and molded into cones 1.75 in. high with 0.5 in. base. The cones were dried in an electric oven, and a series mounted on plaques made of three portions fireclay and one portion Al₂O₃. The plaques of cones were thoroughly dried, and the melting points of the cones determined in accordance with the method for determining the fusion points of coal ash, specified by the American Society for Testing Materials and by the Bureau of Mines. The fusions were made in an air-blast pot furnace, the

plaque of cones being protected from direct contact with the flame by means of a cylindrical muffle. In order to maintain nonoxidizing atmosphere, several lumps of charcoal were placed on the bottom of the muffle. Temperatures were determined by a Leeds & Northrup optical pyrometer, focused on the cones, and readings were taken at the moment the cones began to bend. The determinations were made in triplicate, and the results recorded within the closest 5° C. readings, as given in Table 4, and graphically represented by Fig. 5.

EFFECT OF VARYING COMPOSITION OF SLAGS

The thermal diagram of ferrous oxide + lime + silica mixtures (Fig. 5) shows that certain compositions have lower melting points than others, which suggested that these compositions would be the most favorable to use in calculating composition of slags for the smelting of cassiterite concentrates. Having this idea in mind, the investigation was made in the following order:

1. Effect of varying the composition ($\text{SiO}_2 + \text{FeO} + \text{CaO}$) to produce a "mezzosilicate" slag ($3\text{MO} + 2\text{SiO}_2$), containing from 15 to 50 per cent. FeO, and 42 to 10 per cent. CaO. (Group I.)

2. Effect of varying the composition of the slags by figuring FeO to form monosilicate ($2\text{FeO} \cdot \text{SiO}_2$), and the CaO to form bisilicate ($\text{CaO} \cdot \text{SiO}_2$); the FeO varying from 30 to 55 per cent. and CaO from 28 to 10 per cent. (Group II).

3. Effect of varying the composition of the slags by figuring FeO to form monosilicate ($2\text{FeO} \cdot \text{SiO}_2$), and the CaO to also form monosilicate ($2\text{CaO} \cdot \text{SiO}_2$); the FeO varying from 32 to 60 per cent., and the CaO from 36 to 10 per cent. (Group III).

4. Effect of varying the composition ($\text{SiO}_2 + \text{FeO} + \text{CaO}$) to produce slags of higher (Group IV), and lower silicate degrees (Group V) than the above slags of Groups I and III: the SiO_2 varying from 56 to 24 per cent., the FeO from 25 to 50 per cent., and the CaO from 7 to 36 per cent.

The tin ore used for these tests was roasted Bolivian cassiterite concentrate, crushed to pass through a 20-mesh screen, and roasted at 800° to 850° C. for 3 hr. The roasted cassiterite concentrate had the following composition:

ANALYSIS PER CENT.		RECAST ANALYSIS PER CENT.	
Sn.....	68.50	SnO_2	87.00
Bi.....	1.50	Bi.....	1.5
Fe.....	4.90	FeS	0.35
S.....	0.24	Fe_2O_3	6.55 = 5.94 FeO
SiO_2	1.43	SiO_2	1.43
CaO	0.34	CaO	0.34
Al_2O_3	0.56	Al_2O_3	0.56
Undetermined.....	22.53	Undetermined....	2.27
Total.....	100.00		100.00

Mixtures of 200 g. of the roasted concentrate, a calculated amount of flux, and of anthracite to form CO, by reduction of SnO_2 , were smelted in a gas-fired furnace holding 12 covered crucibles.

The rate and time of heating are shown by the curves of Fig. 6. The metal button was separated from the slag and weighed; being impure, it was liquated in a crucible at low temperature (about $300^\circ\text{C}.$) to remove

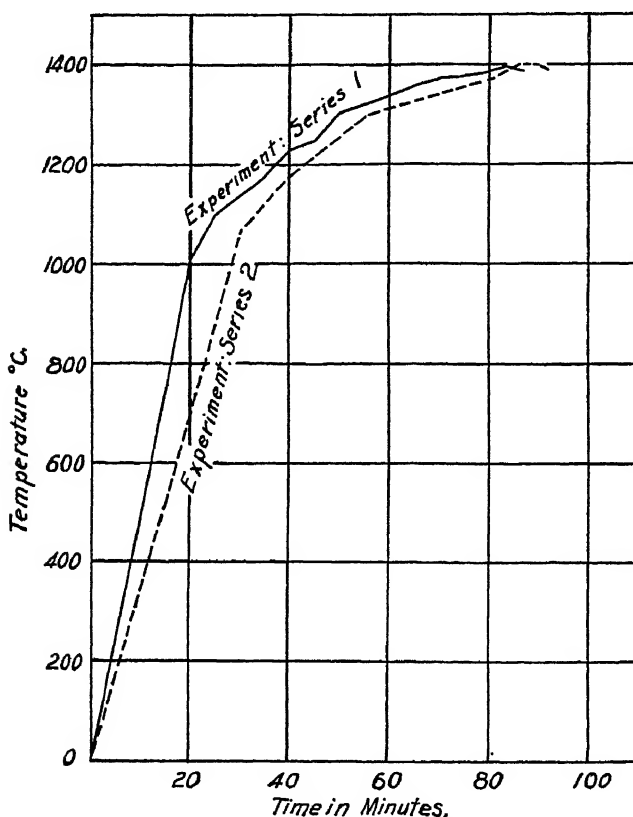


FIG. 6.—TYPICAL RATES OF HEATING OF THE SMELTING CHARGES OF CASSITERITE CONCENTRATES.

most of the impurities, as dross. The liquated metal was poured into a cast-iron mold and reweighed. The surface of the metal, after liquation, was smooth and bright, so this was regarded as pure tin extracted, and percentage extraction calculated.

SERIES I.—PRODUCING A MEZZOSILICATE SLAG

The results of the tests (Series I) are shown in Table 5. The constants and the variables were:

CONSTANTS

"Mezzosilicate" slag (Group I)
 Reducing agent: 10 per cent. excess of anthracite to form CO
 Temperature: 1300° to 1400° C.
 Time at 1300° to 1400° C. = 30 min.

VARIABLES

Slag composition: FeO varied from 15 to 45 per cent., and CaO from 42 to 15 per cent.

TABLE 5.—*Production of a Mezzosilicate Slag*

Charge = Tin Concentrate, 200 Grams; Anthracite, 36 Grams

Mixture No.	Ratio Composition of Slag			Tin Reduced from 200 Parts Ore			Per Cent. Extraction	Remarks
	SiO ₂	FeO	CaO	Crude Tin	Liquated Tin	Dross		
1	43	15	42	a
2	42.5	20	37.5	a
3	42	25	33	a
4	42	30	28	123.0	117.5	5.5	86.5	b
5	41	35	24	128.0	120.5	7.5	88.6	b
6	41	40	19	138.0	119.8	18.2	87.5	b
7	40	45	15	132.0	118.4	13.6	87.0	b

a. The charges were sintered, no slag was formed. Tin was reduced in the form of globules scattered throughout the mass.

b. The charges were completely fused, a small portion of black colored dry residue remained. The separations were very good.

The results of mixtures 1, 2 and 3 show that high lime and low iron oxide produce "mezzosilicate" slags (3 Mo. 2SiO₂) which have too high a formation temperature. For this reason, the high-lime slags were not tried for the subsequent tests. It was also found that a small portion of unfused dry residue remained in the fused charges Nos 4, 5, 6 and 7. This residue was found to be unconsumed anthracite, so the anthracite used for the next set of tests was 98 per cent. of the theoretical amount necessary to reduce the tin oxide and form CO.

SERIES II.—PRODUCING A MEZZOSILICATE SLAG

The results of the next tests (Series II) are given in Table 6. The constants and the variables were:

CONSTANTS

"Mezzosilicate" slag (Group I)
 Reducing agent: 98 per cent. of the theoretical amount of carbon in anthracite to form CO.
 Temperature: 1300° to 1400° C.
 Time at 1300° to 1400° C: 30 min.

VARIABLES

Slag composition: FeO varied from 30 to 45 per cent., and CaO from 28 to 15 per cent.

TABLE 6.—Varying *FeO* and *CaO* of Slags

Charge = Tin Concentrate, 200 Grams; Anthracite, 31.5 Grams

Mixture No.	Ratio Composition of Slag			Tin Reduced from 200 Parts Ore			Per Cent. Extraction	Remark
	SiO ₂	FeO	CaO	Crude Tin, Gm.	Liquated Tin, Gm.	Dross, Gm.		
4	42	30	28	133.9	118.6	15.3	87.3	a
5	41	35	24	135.1	124.0	11.1	91.0	
6	41	40	19	135.2	121.1	14.1	89.0	
7	40	45	15	134.1	120.5	13.6	88.6	

a. The unfused portion of the dry residue was less than before, and the separation of the metal button and the slag were improved considerably.

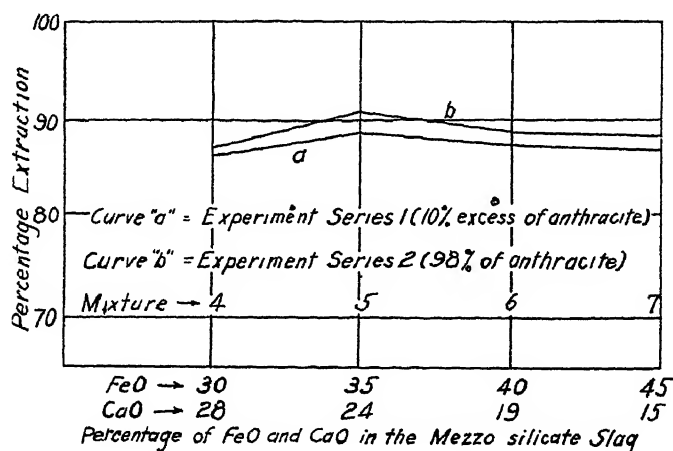


FIG. 7.—EFFECT OF VARIATION OF IRON OXIDE AND LIME CONTENT OF MEZZO-SILICATE SLAGS, AND THE AMOUNT OF ANTHRACITE ON THE EXTRACTION OF TIN FROM CASSITERITE CONCENTRATES.

From the results of the foregoing two series of tests, two curves were plotted (Fig. 7) in order to compare the effects of the variations.

SERIES III.—VARYING THE AMOUNT OF REDUCING AGENT

The two foregoing experiments showed that an excess of anthracite to form CO diminished the tin extraction, principally because of the ash content of the anthracite which tended to increase the viscosity of the slag, and also because excessive reduction led to the formation of an iron-tin alloy. The amount of anthracite in the next set of experiments was decreased to 55 per cent. of the theoretical amount necessary to form CO for reduction of the tin oxide. The results of these tests

(Series III) are shown in Table 7 and Fig. 8. The constants and variables were:

CONSTANTS

"Mezzosilicate" slag (Group I).
 Temperature: 1300° to 1400° C.
 Time at 1300° to 1400° C.: 30 min.

VARIABLES

Reducing agent: Amount of anthracite varied from 10 per cent. excess down to 55 per cent. of the theoretical amount to form CO.
 Slag composition: FeO varied from 30 to 45 per cent., and CaO from 28 to 15 per cent.

A series of curves (4, 5, 6 and 7 of Fig. 8) were plotted from the above experiments, which clearly shows the effect of varying the amount of reducing agent.

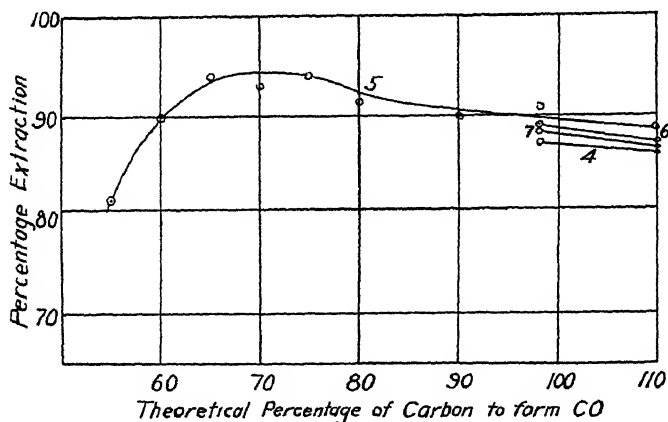


FIG. 8.—EFFECT OF THE AMOUNT OF ANTHRACITE ON THE EXTRACTION OF TIN FROM CASSITERITE CONCENTRATES.

Curve 5 was plotted from tests of Series III, mixture 5; curve 6 from mixture 6; curve 7 from mixture 7; and curve 4 from mixture 4.

These curves make it apparent that an excessive amount of anthracite is not beneficial, as the unconsumed portion of carbon and the amount of ash entering the slag has a tendency to prevent agglomeration of the pellets of tin, thus causing lower extraction of the metal; and also the increase in reducing action causes the formation of iron-tin alloys. The reason for this may be explained by the equilibrium between carbon and its oxides in the preparation of producer gas, which is dependent on temperature: at high temperature very much more carbon monoxide is observed than at low temperature.⁵ From this, the conclusion may be

⁵ R. Schenck and R. S. Dean: *Physical Chemistry of Metals*, 167. J. Wiley & Sons, Inc., New York, 1919.

R. S. Dean: *Theoretical Metallurgy*, 182-184. J. Wiley & Sons, Inc., New York, 1924.

drawn that the formation of carbon monoxide is a process which is accompanied by heat absorption due to the decomposition of the dioxide to monoxide.

Boudouard⁶ determined the composition of carbon monoxide and carbon dioxide mixtures that are in equilibrium with carbon at atmospheric pressure and at different temperatures. His observations and thermodynamic calculations gave 2 per cent. CO at 450° C., 68 per cent. CO at 700° C., 97 per cent. CO at 925° C. and 99 per cent. CO at 1000° C. Above 1000° C. there is only a slight trace of CO₂ in the gaseous phase. The results of the tests of Series III show that less anthracite than the amount theoretically required to form CO gave better extraction of tin than when anthracite was in excess, no doubt due to the formation of some carbon dioxide during the heating of the mixtures up to 1000° C. As the smelting temperature of tin ores is above 1000° C., the reaction atmosphere will be high in CO and low in CO₂, since the former can energetically function as reducing agent only at temperatures between 450° and 900° C.; therefore, the theoretical amount of carbon to form CO would not be necessary, as some CO₂ is produced during the heating of the charge from 450° to 1000° C.

The reducing reactions of stannic oxide by carbon and carbon monoxide with rise of temperature may be expressed as follows:

- I. $C + CO_2 = 2CO$ (450° to 900° C.)
- II. $SnO_2 + CO = SnO + CO_2$ (450° to 900° C.)
- III. $SnO + CO = Sn + CO_2$ (450° to 900° C.)
- $Sn + CO_2 = SnO + CO$ (above 1000° C.)
- IV. $SnO_2 + 2C = Sn + CO$ (above 1000° C.)

The reaction is not a single one, but it may be regarded as progressing in several steps, similar to the iron-carbon reactions in a blast furnace, which are dependent on thermodynamic effect.

SERIES IV.—VARYING THE AMOUNT OF ANTHRACITE AND THE COMPOSITION OF SLAGS

Having determined that only 75 per cent. of the theoretical amount of anthracite to form CO is required for the highest extraction of tin, the next tests (Series IV) were made by using this amount of anthracite, and varying the composition of the slags. The results of these experi-

⁶ L. Guillet: *Traité de Métallurgie Générale*, 229. Baillièrre et fils, Paris, 1922.

ments are shown in Table 8 and by curve of Fig. 9. The constants and variables were:

CONSTANTS

Reducing agent: 75 per cent. of the theoretical amount of carbon in anthracite to form CO.

Temperature: 1300° to 1400° C.

Time at 1300° to 1400° C.: 30 min.

VARIABLES

Varying the amounts of flux require to produce:

"Mezzosilicate" slags (Group I).

(2FeO.SiO₂) + (CaO.SiO₂) slags (Group II).

(2FeO.SiO₂) + (2CaO.SiO₂) slags (Group III).

Slags other than the above (Groups IV, V, and Intermediate).

TABLE 8.—*Varying the Composition of the Slags*

Charge = Tin Concentrate, 200 Grams; Anthracite, 24 Grams

Group	Mixture No.	Ratio Composition of Slag			Flux Added per 100 Parts Ore			Resulting Slag per 100 Parts Ore	Tin Reduced from 200 Grams Cassiterite Concentrate			Percentage Extraction
		SiO ₂	FeO	CaO	SiO ₂	CaO	Total		Crude Tin	Liquated Tin	Dross	
I	4	42	30	28	6.2	5.2	13.4	21.6	131.5	124.5	7.0	91.0
	5	41	35	24	5.0	3.6	8.6	18.6	138.8	128.8	10.0	94.0
	6	41	40	19	3.6	2.5	6.1	15.7	136.5	128.0	8.5	93.5
	7	40	45	15	3.4	1.7	5.1	14.5	136.0	126.3	9.7	92.2
	8	40	50	10	2.9	0.9	3.8	13.1	134.1	125.2	8.9	91.4
II	5	41	35	24	5.0	3.6	8.6	18.6	138.8	128.8	10.0	94.0
	9	38	45	17	3.2	1.9	5.1	14.7	137.2	126.1	11.1	92.1
	10	36	50	14	2.4	1.3	3.7	13.3	137.5	127.0	10.5	92.6
	11	35	55	10	1.9	0.7	2.6	12.0	130.4	123.4	7.0	90.0
III	12	32	32	36	4.2	6.4	10.6	20.4	134.5	126.5	8.0	92.5
	13	32	40	28	2.9	3.8	6.7	16.5	136.7	125.8	10.9	91.8
	14	32	46	22	2.2	2.5	4.7	14.2	138.2	128.2	10.0	93.6
	15	31	52	17	1.7	1.6	3.3	12.7	139.7	127.8	11.9	93.3
	16	30	60	10	1.1	0.7	2.8	11.0	136.9	125.3	11.6	91.5
Intermediate	17	36	34	30	4.4	4.9	9.3	19.2	136.8	126.5	10.3	92.4
	18	36	40	24	3.5	3.2	6.7	16.5	136.6	126.5	10.1	92.4
IV	19	56	37	7	7.5	0.8	8.3	18.0	110.6	106.9	3.5	78.0
	20	55	35	10	7.5	1.4	8.9	18.9	121.6	116.1	4.5	85.0
	21	50	25	25	9.5	5.6	15.1	25.9	122.5	119.3	3.2	87.1
	22	46	36	18	5.7	2.6	8.3	18.3	133.7	125.2	8.5	91.4
	23	45	45	10	4.1	1.0	5.1	14.7	130.3	121.5	8.8	88.8
V	24	30	34	36	3.4	6.0	9.4	19.4	137.6	126.1	11.5	92.1
	25	25	50	25	1.1	2.0	3.7	13.2	140.6	125.9	14.7	92.0
	26	25	45	30	1.4	3.6	5.0	14.7	140.2	125.6	14.6	91.6
	27	24	40	36	1.8	5.0	6.8	16.7	140.1	124.9	15.2	91.1

The analysis of a composite sample of the crude and of the resulting liquated tin was found to be:

	Crude Tin, Per Cent.	Liquated Tin, Per Cent.
Sn.....	94.5 to 96.5	98.50
Bi.....	2.00	1.03
Pb.....	0.17	0.16
Sb.....		0.16
Cu.....		0.15
Fe.....	1.70	nil
Total.....		100

The curve of Fig. 9 was plotted from the results of the tests on variation of slag composition (Series IV), Table 8, by using the relative weight of slag per 100 parts ore smelted as abscissa, and extraction percentage

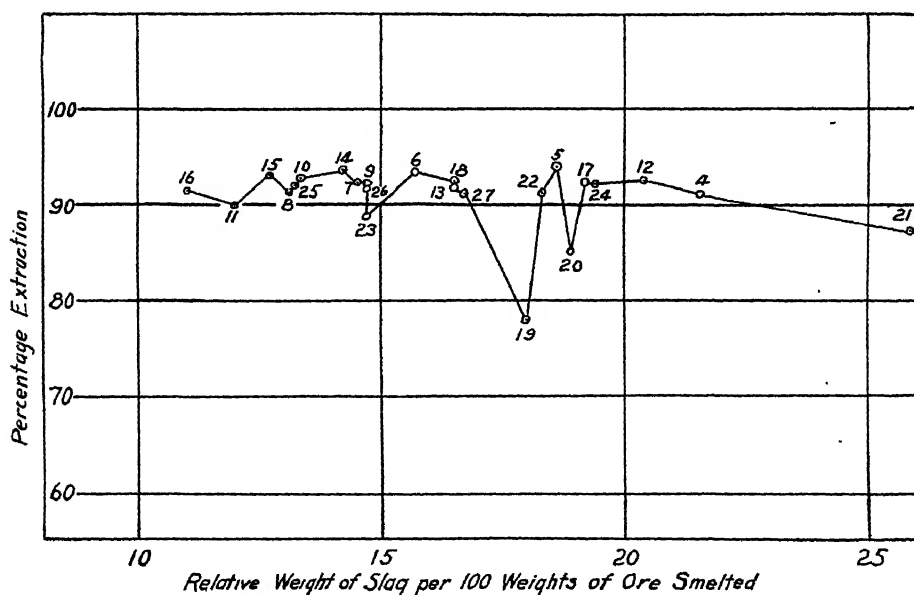


FIG. 9.—EFFECT OF SLAG COMPOSITION AND THE RELATIVE AMOUNT OF SLAG ON THE EXTRACTION OF TIN FROM CASSITERITE CONCENTRATES.

as ordinate, in order to show the effects of the slag variations. This shows that the extraction of tin from cassiterite concentrate by smelting is decidedly influenced by the composition of the slags, and is also dependent on the relative amount of slags produced. The highest extractions

with the production of equal weights of slag were obtained by producing slags whose ratio compositions were as follows:

Group	Mixture No.	Ratio Composition		
		SiO ₂	FeO	CaO
I and II	5	41	35	24
I	6	41	40	19
III	12	32	32	36
III	14	32	46	22
III	15	31	52	17
IV	21	50	25	25

Charge mixtures 5 and 6, which correspond to "mezzosilicates" (3MO.-2SiO₂) with medium FeO and comparatively high CaO contents gave high extraction of tin. and with the production of smallest amount iron-tin dross. Mixtures 14 and 15, corresponding to monosilicates (2MO.SiO₂) of both FeO and CaO, containing lower SiO₂, higher FeO, and slightly lower CaO, gave high extraction of tin, due to lower relative amount of slag produced per 100 parts ore smelted; *i. e.*, less flux was added, as all the charge calculations were based on the FeO content of the cassiterite concentrates; no iron oxide flux was added in any case. Very low extractions occurred when FeO content of the slags was high, and CaO below about 14 per cent., as shown by mixtures 8, 11, 16, 19, 20 and 23. In these cases the extraction was considerably lower than for the mixtures which produced corresponding weights of slags, due in case of mixtures 8, 11 and 16 to the very low CaO content of the resulting slags, and in case of mixtures 19, 20, 21 and 23 to the slags being very siliceous. Such slags dissolve stannous oxide and thus decrease the extraction. The mixtures which contained the lower SiO₂ and higher lime gave higher extractions, as demonstrated by Nos. 12, 13, 17 and 24; but it should not be overlooked that the lower the silicate degree of the slag produced, the larger the amount of iron reduced, which increased the amount of iron-tin dross, as shown by the difference of the weights of the crude tin and liquated tin (Table 8).

The results in Table 8, and also the curve of Fig. 9 suggested that in the treatment of cassiterite concentrates, or tin ores, it will be advantageous to calculate the smelting mixture with regard to the analysis of the ore to be treated, and not attempt to produce a definite type of slag. The curve of Fig. 9 will serve as a guide for the selecting of the most suitable ratio composition of the slag, no matter what the analysis of the concentrates may be. If the concentrate or the ore contains high tin and low iron, it will be better to calculate the ratio composition of the slag to be similar to that of mixtures 5 and 6 (Group I); whereas if the tin ore contains low tin and high iron, it will be advisable to select the ratio

composition of slag similar to mixtures 14 and 15 (Group III), in order to produce relatively smaller amounts of slag per 100 parts of ore smelted, and thus diminish the tin loss in the slag. In case the cassiterite concentrates contain high SiO_2 and low FeO , the most advantageous smelting mixture to select would no doubt be that corresponding to the ratio composition of No. 21 (Group IV).

The above suggestion was tried out, and it was found that the results were as anticipated. The tin ore used was flotation concentrate of a low-grade ore from the Kotechu Tin Trading Company's mine. The ore was first treated by a combined flotation and leaching process, which raised the tin content from 23.1 per cent. Sn, to 55.4 per cent. Sn, and contained 0.4 per cent Pb, 2.1 per cent. SiO_2 , 14.0 per cent. Fe, and 1.2 per cent. CaO . The first test was made by the "direct-reduction" method with anthracite, at about 900°C . by the stirring operation, without addition of flux. Tin and iron were reduced together and formed "hard-head" from which no tin could be liquated.

The second test was made by smelting mixtures of the concentrates and flux, which were calculated to produce slags of the ratio composition similar to mixture 5 (Group I) and mixture 15 (Group III); these were smelted under the same conditions as tests of Series IV. The results are given in Table 9.

TABLE 9.—*Varying Amounts of Flux*

Test No.	Ratio Composition of Slag			Tin Concentrate, Grams	Anthracite, Grams	Flux Added Per 100 Parts Ore			Resulting Slag Per 100 Parts Ore	Tin Reduced, Grams			Percentage Extraction
	SiO_2	FeO	CaO			SiO_2	CaO	Total		Crude Tin	Liquated Tin	Dross	
1	Direct reduction without producing slag			100	20	No flux added				64.0	0	65	
2	41	35	24	150	17	18.0	9.7	27.7	54.6	61.5	58.2	3.3	70.0
3	31	52	17	150	17	8.4	4.4	12.8	38.2	76.2	65.8	10.4	80.0

It is evident that this kind of concentrate cannot be treated by the "direct-reduction" process, on account of its high iron content (equivalent to 18.0 per cent. FeO); but it can be smelted by selecting a suitable slag ratio composition with the aid of the results of the tests of Series IV, and the curve of Fig. 9. A higher tin extraction from the Kotechu concentrate may possibly result from a preliminary heating with carbon at 700° to 750°C . in order to render the iron magnetic⁷ followed by

⁷ Without the presence of carbon, Fe_2O_3 is not rendered magnetic, unless it is heated between 1100° and 1200°C . (Results of tests previously made.)

removal of the larger part of the iron by magnetic concentration. The resulting final tin concentrate can then be smelted to better advantage. The above tests were made merely to illustrate the advantage of selecting the smelting mixture with regard to the analysis of the ore to be treated, rather than depending on producing a definite type of slag.

SERIES V.—EFFECT OF VARYING AMOUNT OF SLAG

In some smelters, old slags are often added to a smelting charge for the purpose of causing more rapid fusion, but unless an increase in weight of metal recovery results, this practice is of no advantage.

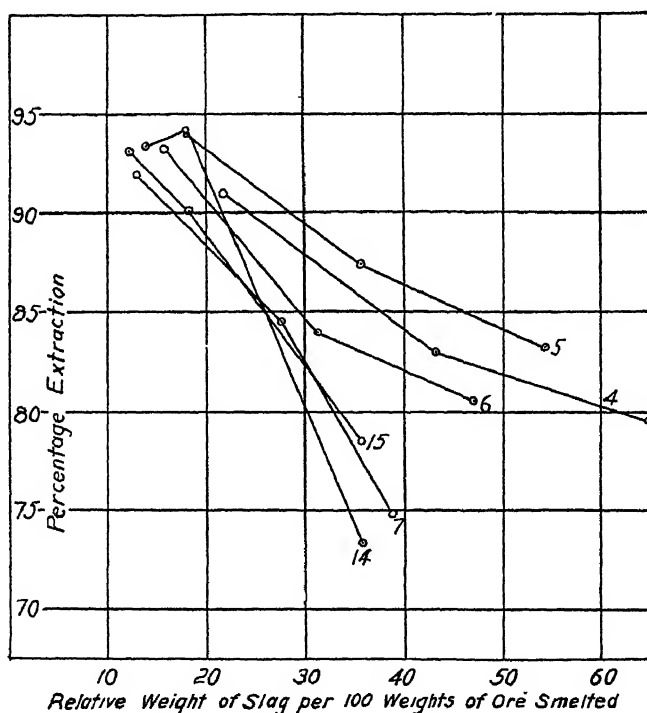


FIG. 10.—EFFECT OF THE RELATIVE AMOUNT OF THE RESULTING SLAGS ON THE EXTRACTION OF TIN FROM CASSITERITE CONCENTRATES.

The effect of varying the relative amount of slag was determined by tests of Series V. The first variations were made by using the minimum amount of flux required to produce the most desirable types of slags, as determined by tests of Series IV. The next variation was made by increasing the relative amounts of slag per 100 parts of ore smelted, keeping the composition of slag in the same SiO_2 : FeO : CaO ratio. The results of

these tests are given in Table 10 and curves of Fig. 10. The constants and variable were:

CONSTANTS	VARIABLE
Composition of slags.	Relative amount of the resulting slags per
Reducing agent: 75 per cent. of theoretical amount of anthracite to form CO.	100 parts ore smelted.
Temperature: 1300° to 1400° C.	
Time at 1300° to 1400° C.: 30 min.	

The results of the tests given in Table 10 and shown by the curves of Fig. 10, demonstrated that the larger the amount of slag produced the smaller is the extraction, even though the mixtures gave slags possessing perfect fluidity. These tests showed also that tin was lost in the slag, and by the formation of an increased amount of iron-tin dross.

In the case of mixture 14 (Group III), when the relative amount of slag was slightly increased from 13.9 to 18.1 parts per 100 parts ore smelted, it showed a slightly higher extraction, due probably to a better separation of the suspended tin pellets from the slightly larger volume of fluid slag.

As SnO_2 is amphoteric, it can act either as a base or as an acid constituent of slag.⁸ If combined with silica, it forms a difficultly fusible silicate; whereas if it combines with lime, calcium stannate may be formed. In either case, loss of tin occurs, so the amount of fluxes used should be just sufficient to produce a suitable slag which possesses the necessary fluidity to permit rapid settling of the reduced pellets of tin.

REDUCING POWER OF THE VARIOUS SOLID CARBON REDUCING AGENTS

The solid reducing agent used for the foregoing series of experiments (Series I-V) was anthracite, which was chosen first because of its high fixed carbon, low volatile matter, and its density. As anthracite is not available in some localities, the reducing power of the various solid reducing materials was compared with that of anthracite. The composition of the reducing agents tested are shown in Table 11.

TABLE 11.—*Composition of Reducing Agents*

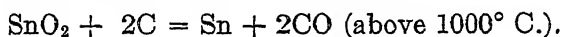
Solid Reducing Agent	Moisture, Per Cent.	Sulfur, Per Cent.	Ash, Per Cent.	Volatile, Per Cent.	Fixed Carbon, Per Cent.
Anthracite.....	1.5	1.3	7.2	2.8	87.2
Bituminous coal...	2.2	1.1	10.2	11.8	74.7
Charcoal.....	6.4		1.5	11.4	80.7
Coke.....	0.2		8.3	0.4	91.1

The tests made for determining the reducing power of solid fuels on tin concentrates were carried out in the same manner as the smelting tests.

⁸ W. Gowland: *The Metallurgy of the Non-ferrous Metals*, 518. Griffin, London, 1921.

The charges of roasted cassiterite concentrates and flux were figured so as to produce slags having the ratio composition 41SiO_2 : 35FeO : 24CaO (mixture 5, Group I). The amount of anthracite used was 75 per cent. of the theoretical amount required to form CO , which was found by tests Series III and IV (Figs. 8 and 9) to give the highest extraction of tin. The same proportionate amount of bituminous coal was used. With charcoal and coke, the amount used in the two first tests was 75 per cent. of the theoretical amount required to form CO , based upon the fixed carbon contents, but for the second test of coke, 34 per cent. excess was used. The temperature was brought up to and held at 1300° to 1400° C. for one-half hour, so as to obtain a good fusion. The results of these tests (Series VI) are given in Table 12.

Theoretically, one part of pure carbon reduces 4.9 parts of tin from stannic oxide in accordance with the reaction:



Bituminous coal in test No. 3, Series VI gave approximately the same reduction value: 1 part having reduced 5.15 parts Sn, whereas anthracite gave a value of 5.36. When an excess of anthracite or coke was used, the excess prevented the collection of the metal pellets into a single mass, and also caused the formation of iron-tin alloy. Both charcoal and coke showed poor reducing power for the reduction of cassiterite concentrate.

CONCLUSIONS

The experiments may be divided into two main parts: (1) the "direct-reduction" of roasted cassiterite concentrates without the use of flux for producing slags; and (2) the smelting of roasted cassiterite concentrates with flux to produce desirable slags.

"Direct-reduction" of Roasted Cassiterite Concentrates without the Use of Flux

Roasted cassiterite concentrate is readily reduced by means of solid carbonaceous fuels and without the addition of flux at temperatures between 850° and 900° C., the latter temperature causing more rapid reduction. At about 800° C., the reduction is slow; at about 760° C. reduction is not apparent; above 1000° C. the temperature is too high, causing reduction of much iron, and sintering of the constituents of the charge, which prevents complete reduction of the tin and retards its collection as a single mass, unless a retarder (usually lime) is added to prevent sintering of the gangue.

Stirring the charge during "direct-reduction" between 850° C. and 900° C. caused rapid and complete collection of the scattered tin pellets

TABLE 12.—*Reducing Power of Solid Fuels*
Tin Concentrate Used = 200 Grams

Test No.	Reducing Agent	Ratio Composition of Slag		Reducing Agent, Grams	Tin Reduced, Grams		Percentage Extra-ction	Reducing Power: Weight of Tin Reduced by 1 Weight of Reducing Agent	Relative Reducing Power at 1300° to 1400° C.	Remarks
		SiO ₂	FeO		Crude	Liquated				
1	Anthracite	41	35	24	137.3	128.0	93.4	5.36	1	Small portion of unfused dry residue and carbon remained.
		41	35	24	139.2	129.5	94.4			
2	Bituminous coal	41	35	24	135.7	126.3	92.2	5.27	0.982	Slag was quite glassy and black in color, metal and slag separation good.
		41	35	24	133.6	126.8	92.5			
3		41	35	24	141.2	128.5	94.0	5.15	0.96	Slag was quite glassy and black in color, metal and slag separation good.
		41	35	24	138.5	127.3	93.0			
4	Charcoal	41	35	24	113.5	111.2	81.2	4.22	0.786	Slag was brown in color, less glassy than 2 and 3, completely fused, and no dry residue.
		41	35	24	110.5	108.0	78.9			
5		41	35	24	113.7	109.5	80.0	3.94	0.735	Same as 4; only slight excess charcoal was used to form CO.
		41	35	24	105.5	100.8	73.8			
6		41	35	24	105.5	100.1	73.0	4.45	0.830	Slag was completely fused, no dry residue.
		41	35	24	109.6	104.5	76.4			
7	Coke	41	35	24	120.2	106.0	77.4	3.17	0.592	Comparatively large amount of dry residue remained on surface of the center portion of slag. 34 per cent. excess coke was used to form CO. Large amount of iron-tin dross produced.
		41	35	24	124.0	107.5	78.5			

into a single mass and enabled a clean separation of the reduced metal from the dry gangue residue, with reduction of smallest amount of iron.

Cassiterite concentrates containing about 20 per cent. Fe_2O_3 and low tin (about 55 per cent. Sn) can not be treated by the "direct-reduction" process, as there is a tendency to produce an iron-tin alloy from which tin will not separate by liquation.

Smelting of Roasted Cassiterite Concentrate with Flux to Produce Suitable Slags

The amount of anthracite or bituminous coal necessary for efficient reduction of roasted cassiterite concentrates is less than the theoretical amount to form CO; between 65 and 75 per cent. of the theoretical amount of carbon required to form CO gave the best tin extraction. When more than this amount was used, an iron-tin alloy was produced; and conversely, when less than this amount was used, tin oxide entered the slags: in both cases lower reduction efficiency resulted. The proper amount of carbon required in order to obtain the highest tin extraction is that which will yield the smallest amount of iron-tin alloy, and also prevent SnO entering the slag.

The best slags to produce in the smelting of roasted cassiterite concentrates were found to be those of Table 13.

TABLE 13.—*Desirable Slags in Smelting Roasted Cassiterite Concentrates*

Group	Mixture No.	Ratio Composition		
		SiO_2	FeO	CaO
I and II	5	41	35	24
I	6	41	40	19
III	14	32	46	22
III	15	31	52	17

Slags corresponding to mixtures 5 and 6, with medium FeO and comparatively high CaO contents, prevented excessive reduction of iron and also prevented tin from entering the slag, thus causing high extraction of tin and production of small amount of iron-tin dross. Slags corresponding to mixtures 14 and 15, with lower SiO_2 , higher FeO, and lower CaO contents, gave high extraction of tin, due principally to lower relative amount of slag produced per 100 parts ore smelted. Slags containing high FeO, with the CaO ratio below about 14, corresponding to mixtures 8, 11 and 16 (Table 8) gave lower extraction of tin, due to the very low CaO content of the resulting slags, which was not sufficient to prevent tin from entering the slag. Slags with the SiO_2 ratio above 45, and CaO below 10, corresponding to mixtures 19, 20 and 23 (Table 8) gave very

low extraction of tin, because they were siliceous and low in CaO, and therefore dissolved stannous oxide and decreased the tin extraction. The slags that contained lower SiO_2 and higher lime, corresponding to mixtures 26 and 27 (Table 8) gave higher tin extractions than when the SiO_2 was high and the lime low (compared with mixtures 19 and 20). The lower the SiO_2 , and the higher the FeO content in the slags, the larger the amount of iron reduced, which increased the amount of iron-tin dross causing decreased tin extraction.

Cassiterite concentrates containing high iron (about 20 per cent. Fe_2O_3) and low tin (about 55 per cent. Sn) can be smelted by producing a slag similar to mixture 15. In this case a relatively smaller amount of slag is produced per 100 parts of ore smelted, which decreases the tin losses in the slag. The selection of a smelting mixture similar to 15 also minimizes the amount of barren flux to be used.

The amount of fluxes used should not be more than necessary to produce the most desirable slag, the fluidity of which will permit rapid settling of reduced tin. An excess of flux or the addition of large amounts of old slag causes loss of metallic tin. Old slags that are high in tin may be smelted separately by the addition of sufficient lime to form a fluid slag, containing between 18 and 30 per cent. CaO and from 32 to 36 per cent. SiO_2 , and using the proper amount of carbon, so as not to form an excess of iron-tin alloy.

DISCUSSION

E. F. KERN, New York, N. Y.—This paper reports the work of two students, Mr. Gill and Mr. Loo. So far as the paper is concerned, it is presented as student's work and not with an idea of revolutionizing the tin industry.

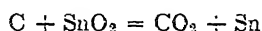
We undertook this work to prove out certain things in the matter of reaction temperature, effect of slag composition, variation in amount of reducing agent, and the effect of different reducing agents.

As to the actual metallurgy of tin, we feel that the work of H. H. Alexander stands foremost of any work that has been done in smelting and refining of tin, so we presented an abstract of Alexander and Stack's paper with that idea, and because it gave us suggestions for undertaking the work of proving certain things that were done by Alexander, which were not done by other metallurgists.

We dwell upon Fig. 9 because it will enable the ready selection of what we call desirable slags. It substantiates our thesis that it is essential to select certain slags in order to obtain desirable results in the smelting of tin ore which contain varying amounts of slagable constituents.

G. E. DALBEY, East Berlin, Conn.—In trying to reduce some tin hydroxide, I had difficulty in smelting in a small blast furnace, but have had considerable success in reducing it in a cylindrical graphite retort externally fired. I introduced the tin hydroxide into the retort with enough carbon for a slight excess over the required amount for complete reduction of the tin to form CO. The excessive carbon seriously interfered with the collection of the tin in one large mass. All the tin was reduced, but the great portion remained as small pellets distributed throughout the residual

carbon. I then tried reducing the amount of carbon so that the reaction would be as follows:



Practically all the tin was reduced and collected in one large mass. Have you any results on excess carbon?

E. F. KERN.—Yes, an excessive amount of carbon will prevent the collecting together of the tin pellets. By the direct reduction method it is a case of collecting the pellets into a large mass by stirring. The apparatus we prefer for this purpose would be a rotary horizontal furnace of the Bureau of Mines, described in a paper by Williams, Barrett and Larsen.⁹ This type of furnace may be considered a muffle furnace, with the CO_2 acting as a muffle over the top of the reducing charge and the excess carbon collecting over the top of the rolling charge, thus preventing oxidation of the reduced tin.

W. CAMPBELL, New York, N. Y.—The ternary system CaO-FeO-SiO_2 is of great interest, not only as bearing on tin and other non-ferrous slags, but also on the basic open-hearth slags in steelmaking.

The Hofman-Babu¹⁰ diagram, unfortunately, has not been studied in the light of modern work on the silicates. Hofman's paper on the temperatures at which certain silicates are formed was presented to the Institute¹¹ in 1899. His results were discussed by Ashley¹² in 1901. Babu¹³ gave the ternary diagram in 1904.

An interesting comparison might be made between this diagram and the one just presented by Loo and Kern, and the question naturally comes up as to why these very marked differences. The answer must be that we are dealing with a four-constituent and not a three-constituent system, because Fe_2O_3 is usually present. Of course, discrepancies must occur when one of the constituents is FeO and the atmosphere is not controlled. This is shown clearly when we consider the work already done on the binary system FeO and SiO_2 . Whiteley and Hallimond¹⁴ in 1919 gave the diagram shown in Fig. 11. There are two eutectics, that of FeO and 2FeO-SiO_2 at 24 per cent. SiO_2 and 1240°C . and that of 2FeO-SiO_2 and SiO_2 at 40 per cent. SiO_2 and 990°C . The freezing point of 2FeO-SiO_2 is about 1280°C . Keil and Dammann¹⁵ in 1925 made melts in an atmosphere of nitrogen. The FeO used contained 7 per cent. Fe_2O_3 , however. Their diagram up to 45 per cent. FeO is shown in Fig. 12. Again we have two eutectics, but the first occurs at 4 per cent. SiO_2 and 1075°C . While the composition of the second is still 40 per cent. SiO_2 , the temperature is 1115°C . More striking still is the melting point of Fayalite, 2FeO-SiO_2 , at a little above 1500°C . as compared with 1280°C . of Whiteley and Hallimond and 1185°C . of Loo and Kern.

⁹ See U. S. Bur. Mines Bull. 270 (1927) 114.

¹⁰ H. O. Hofman: General Metallurgy, 458. 1913.

¹¹ H. O. Hofman: Temperatures at Which Certain Ferrous and Calcic Silicates are Formed in Fusion. *Trans.* (1899) 29, 682.

¹² H. E. Ashley: Slag Constitution, Studied by Means of the Tri-axial Diagram with Rectangular Coordinatities. *Trans.* (1901) 31, 855.

¹³ L. Babu: *Traité théorique et pratique de métallurgie générale*, 1. Paris, 1904-6. Bérauger.

¹⁴ J. H. Whiteley and A. F. Hallimond: The Acid Hearth and Slag. *Jnl. Iron and Steel Inst.* (1919) 99, 199.

¹⁵ O. v. Keil and A. Dammann: Beitrag zur Kenntnis des Zustandsdiagramms Eisenoxydul und Kieselsäure. *Stahl und Eisen* (1925) 45, 890.

Furthermore, the recent work of Greig¹⁶ proves that the system $\text{FeO} \cdot \text{SiO}_2$ shows two liquids containing about 59 and 96 per cent. SiO_2 respectively at 1690°C. and we have the separation into two layers (Fig. 13).

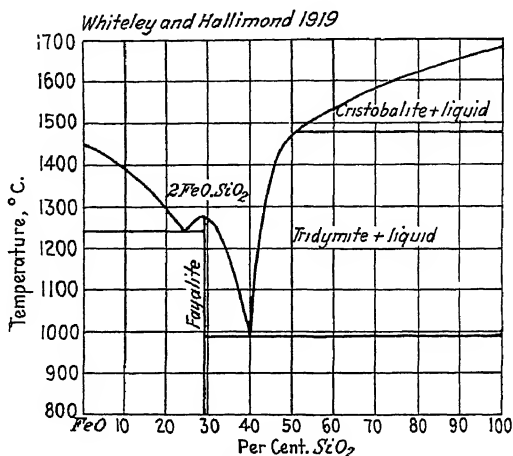


FIG. 11.

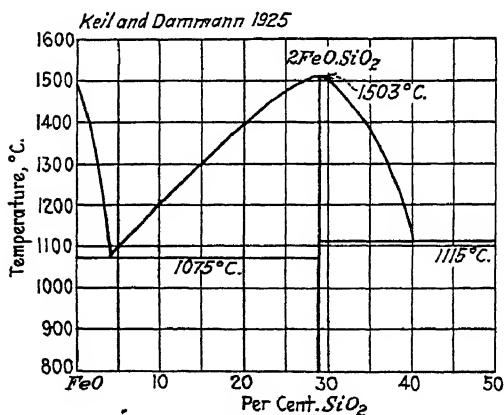


FIG. 12.

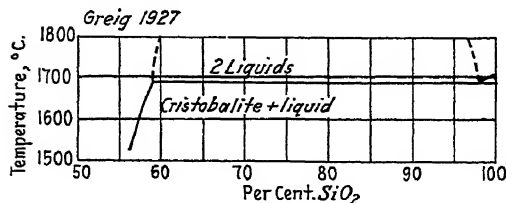


FIG. 13.

It seems unfortunate that we still cling to the old nomenclature and speak of mezzosilicates and the like, when referring to slags which are obviously mechanical

¹⁶ U. W. Greig: On Liquid Immiscibility in the System $\text{FeO} \cdot \text{Fe}_2\text{O}_3 \cdot \text{Al}_2\text{O}_3 \cdot \text{SiO}_2$. *Amer. Jnl. of Sci.* (1927) 14, 473.

mixtures, as shown by Vogt in his work on silicates in 1903 and so clearly demonstrated by the wonderful diagrams from the Geophysical Laboratory, Washington, and by those of McCaffery and others. Thus the FeO-SiO_2 diagram shows only one compound, $2\text{FeO} \cdot \text{SiO}_2$ or Fayalite, melting point or formation point or fusion point given as 1500°C. , by Keil and Dammann. Fayalite forms eutectics with FeO and with SiO_2 . The composition corresponding to $3\text{FeO} \cdot 2\text{SiO}_2$, or 40 per cent. SiO_2 , is the eutectic of Fayalite and SiO_2 . The composition corresponding to $\text{FeO} \cdot \text{SiO}_2$, or 45.5 per cent. SiO_2 is a mechanical mixture of SiO_2 and the eutectic and therefore begins to melt at the eutectic temperature.

The CaO-SiO_2 diagram is too well known to need an illustration. It shows two compounds on the liquidus; viz, $\text{CaO} \cdot \text{SiO}_2$ at 1540°C. and $2\text{CaO} \cdot \text{SiO}_2$ at 2130°C. There are three eutectics (see *e, g, and k, Fig. 14*).

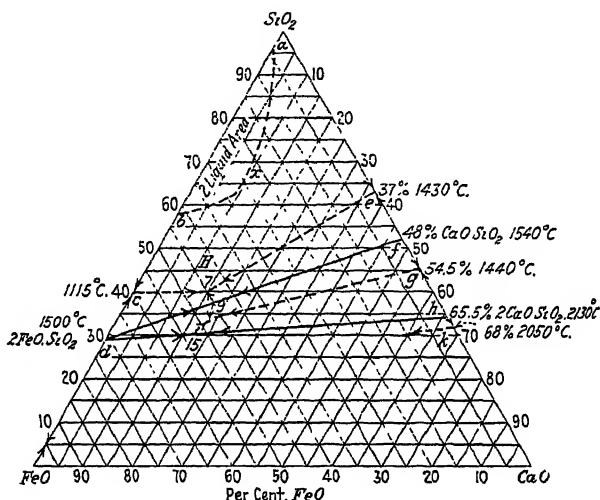


FIG. 14.

Using these data on a ternary diagram and applying the results obtained by Loo and Kern, we can assume two ternary systems:



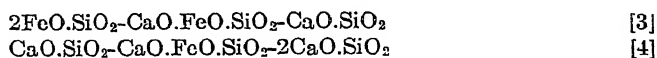
The system $2\text{FeO} \cdot \text{SiO}_2 - \text{CaO} \cdot \text{SiO}_2 - \text{SiO}_2$ will have a two-liquid area $a \times b$, whose boundary has yet to be determined. The ternary eutectic is indicated at 7, in Fig. 14, about 40 per cent. SiO_2 , 45 per cent. FeO , 15 per cent. CaO at 1120°C. , or slag No. 7 of Loo and Kern.

Hofman shows a minimum point at *H* at 1030°C. and 45 per cent. SiO_2 , 45 per cent. FeO , 10 per cent. CaO , which corresponds to the present authors' slag No. 23 at 1195°C.

The system $\text{CaO} \cdot \text{SiO}_2 - 2\text{FeO} \cdot \text{SiO}_2 - 2\text{CaO} \cdot \text{SiO}_2$ shows minima at 9, the eutectic of $2\text{FeO} \cdot \text{SiO}_2$ and $\text{CaO} \cdot \text{SiO}_2$ and at 15 the eutectic of $2\text{FeO} \cdot \text{SiO}_2$ and $2\text{CaO} \cdot \text{SiO}_2$. Perhaps then somewhere between them would occur a ternary eutectic melting about 1120°C. , here given as 34 per cent. SiO_2 , 47 per cent. FeO , 19 per cent. CaO .

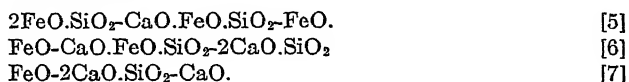
On the other hand, when we use the Hofman-Babu diagram, the line from $2\text{FeO} \cdot \text{SiO}_2$ to $2\text{CaO} \cdot \text{SiO}_2$ shows a second minimum at about 36 per cent. CaO , 32 per cent. SiO_2 , 32 per cent. FeO , which would indicate that the system does not belong to Roseboom's type V. There is reason to suspect a compound $\text{CaO} \cdot \text{FeO} \cdot \text{SiO}_2$ forming

eutectics with $2\text{FeO} \cdot \text{SiO}_2$ and $2\text{CaO} \cdot \text{SiO}_2$. If this is true, we are dealing with two ternary systems:



We might speculate further as to the phases in the area $\text{FeO} - 2\text{FeO} \cdot \text{SiO}_2 - 2\text{CaO} \cdot \text{SiO}_2 - \text{CaO}$. Hofman's work shows a slight maximum at about 17 per cent. CaO , 24 per cent. SiO_2 , 59 per cent. FeO and a minimum at about 27 per cent. CaO , 21 per cent. SiO_2 , 52 per cent. FeO on a line drawn from CaO to $2\text{FeO} \cdot \text{SiO}_2$. The question arises whether we are dealing with a binary of CaO and $2\text{FeO} \cdot \text{SiO}_2$ or of FeO and $2\text{CaO} \cdot \text{SiO}_2$.

It would appear that melts high in line show FeO , $2\text{CaO} \cdot \text{SiO}_2$ and CaO but no $2\text{FeO} \cdot \text{SiO}_2$, also that FeO and CaO form a eutectic. With these meager data, one might expect to find three ternary systems:



but until this region of the diagram is thoroughly explored, this must remain guesswork.

The effect of Fe_2O_3 (and Fe_3O_4) on this system $\text{FeO} - \text{CaO} - \text{SiO}_2$ is not simply contamination, it is to produce a quaternary system, needing a three-dimensional model like that of the $\text{CaO} - \text{MgO} - \text{Al}_2\text{O}_3 - \text{SiO}_2$ to explain it and the reason why the work of two authors does not agree is because their results are not for the same plane through the quaternary model.

If pressure could be brought to bear on the Geophysical Laboratory perhaps they would work out the system $\text{FeO} - \text{CaO} - \text{SiO}_2$ the way they have done the lime, magnesia, alumina, silica systems and then we should have a sound basis from which to begin our work on slags containing iron oxide.

In conclusion I would like to add my thanks to Messrs. Loo and Kern for their painstaking work on this most interesting series.

E. F. KERN.—I received a letter from Mr. Mulligan, of East Chicago, who brought up the matter of tin losses, and asked whether the work was done with cassiterite containing sulfur or sulfide. I think that this brings up the point of which Mr. Linville has spoken, as to the tin losses.

We know that when sulfur is present, most of the sulfur will combine with tin to form tin sulfide, which has a low volatilization point, about 900°C . We tried to eliminate as many variables as possible in conducting the investigation, so did not introduce sulfur. We are very glad that we stimulated thought that will no doubt awaken more research along this line.

Metallurgical Treatment of Flotation Concentrates

BY ARTHUR S. DWIGHT,* NEW YORK, N. Y.

(New York Meeting, February, 1928)

THE flotation process of ore concentration has been a great boon to the miner but to the metallurgist it has not been an unmixed blessing. Fortunately for him, however, the development of the process has been gradual, so that the problem of smelting large proportions of flotation concentrates did not arise until methods for doing so successfully had been developed.

PROBLEMS IN SMELTING LEAD FLOTATION CONCENTRATES

Taking up first the lead field, flotation concentrates offer serious difficulties in handling and in preparing for the blast furnace, quite aside from the problems presented to the furnaceman by the necessity of smelting smaller tonnages of material higher in grade and usually basic in character, with the consequent changes in flux requirements and furnace operation. As received at the smelter, these concentrates are always wet and usually of the consistency of sticky mud. Difficult enough to handle in warm weather, they are worse in winter, when it may be necessary to blast them out of the cars. They hang up in bins and chutes and stick to conveyors, requiring the constant attention of the operators. If dried they become like wheat flour, involving heavy dust losses in handling and in furnace operations.

Roasting in hearth furnaces is more difficult than with coarser materials. Dust losses are increased and, what is more serious, the moist concentrates tend to form balls or nodules, which harden and pass almost unaltered through the furnace with consequent high sulfur in the product, while the unballied fine particles lie on the hearth in a dense bed, practically impenetrable to the oxidizing gases, so that roasting takes place only on the surface of the bed. A more prolonged roasting period is therefore required and the capacity of a given furnace is decreased.

They are entirely too fine to be charged into the blast furnace, even if thoroughly roasted; excessive flue-dust losses, irregular operation and slowing up of the furnace would be the obvious result. Consequently, it was early recognized that flotation concentrates would have to be put in coarser condition for smelting and the universal method adopted has been the Dwight & Lloyd process of sintering. This did not usually

* President, Dwight & Lloyd Companies.

require any change in equipment, since most of the lead smelters were already provided with sintering machines, as the Dwight & Lloyd process had become standard practice even before flotation concentrates were being received in appreciable tonnages.

At first the tonnages of flotation concentrates shipped to the smelter were small, the bulk of the ore receipts being crude ores and coarse concentrates. The small proportion of flotation could be mixed in with this coarser material without upsetting the routine methods, although it might require partial drying and otherwise entail some additional expense in handling. A small percentage of flotation would be mixed with the coarser sulfides, the mixture preroasted, if necessary, and then sintered in the usual way. Although there was somewhat more flue dust made, especially when preroasting in multiple-hearth furnaces, the proportion of excessively fine material in the mixture was so low as not to interfere seriously with the operation. When sufficient coarse materials are available, this method is often followed now, but not many smelters are today so fortunate as to receive much of their ore in coarse form.

The development of selective flotation has resulted in changing still further the treatment necessary for preparing the concentrates for the blast furnace. Formerly it was usual, in lead smelting, for the blast-furnace charges to contain sufficient copper to make desirable a considerable matte fall, and the roasting and sintering operations were conducted with a view to leaving 3 to 5 per cent., or even more, sulfur in the sinter. But as selective flotation became more generally practiced, less copper was recovered in the lead concentrate, so that a smaller matte fall became desirable; in extreme cases, no matte at all was made but the small amount of copper was collected as dross in the bullion. Therefore it was desirable that less sulfur be charged into the blast furnace, and this necessitated making a sinter of lower sulfur content.

Double Sintering

Double sintering affords a means of accomplishing this, and as it also facilitates the treatment of flotation concentrates, it is extensively practiced. This consists in blast-roasting the ore twice, making in the first pass or "first over" a preroasted product containing a small amount of agglomerate (fused sulfide); this is crushed and the entire charge is given a second pass over the machine, the result being a strong, low-sulfur sinter cake. No attempt is made to produce sinter in the first pass, as its purpose is primarily a preroasting operation. The sintering machine is run at a higher speed and usually with a deeper bed. The first pass reduces the sulfur to 9 to 12 per cent. When crushed to minus $\frac{1}{2}$ or $\frac{1}{4}$ in. and again moistened, the fuel values contained in this product are such that a sinter running 1.5 to 2 per cent. sulfur and of ideal physical character is produced in the second pass.

Another great advantage of this treatment, so far as sulfur elimination is concerned, results from crushing the ore between the two sintering steps. As noted, wet flotation concentrates have a decided tendency to build up into balls or nodules, from pin head to 1 or 2 in. size. The smaller nodules are in ideal physical condition for sintering, but the larger ones bake hard and are difficult to desulfurize. If sintered without being crushed beforehand, they retain their shape and pass through the operation with surface fusion and only slight sulfur elimination, so that the finished product carries a higher proportion of sulfur than would otherwise be the case. Crushing the "first over" product breaks these nodules with the rest of the material and reduces them to the proper size for feed to the "second over" operation, which consequently produces a low-sulfur sinter.

Flotation concentrates have one characteristic that is helpful to the metallurgist who has them to sinter; *i. e.*, a lower sulfur sinter can be made from them than from coarse material of the same original chemical composition. For example, starting with 30 per cent. sulfur in the charge in each case, coarse sulfide ore crushed to $\frac{1}{4}$ in. may roast in the first pass over the sintering machine to perhaps 18 to 20 per cent. sulfur; when crushed to 8 or 10 mesh the product will be 2 or 3 per cent. lower; but if the charge consists of flotation concentrates the product will probably run 12 to 15 per cent. sulfur. This is due to the fact that the finer the particles of sulfide, the more rapid the oxidation by the air currents, provided of course that the air currents can reach the individual particles. The very nature of flotation concentrates renders this difficult, but it can be done.

ESSENTIAL PRINCIPLES OF ROASTING AND SINTERING BY INTERNAL COMBUSTION

It will help to make the matter clear if we review for a moment the essentials of the Dwight & Lloyd process of roasting and sintering by internal combustion. As ordinarily practiced, it consists in arranging the ore on a grate in a thin layer, uniformly pervious to air currents, igniting the combustible elements on the upper surface of the mass and passing air currents through the layer in a downward direction. The ignition thus started on the upper surface is slowly propagated downward in a thin zone of intense oxidation producing at a given point a momentary melting effect, at which moment the semifused material is whipped by the air currents into thin films and cells and the next instant is chilled into a porous cokelike mass we call "sinter." The zone of reaction moves slowly downward through the mass until it reaches the grate, and the sinter cake is finished and ready to be discharged. About 15 min. is required to sinter a 4-in. layer. At any stage of the procedure, the stratum above the zone of reaction will be finished sinter while below it

will be raw charge. The advantages of sintering in connection with many metallurgical operations have been so positively demonstrated that they need not be referred to here.

The peculiar structure of sinter is shown in Fig. 1, which shows the polished surface of a sinter cake made from galena. The cell walls, actually $\frac{1}{16}$ to $\frac{1}{400}$ in. thick, offer an incomparably greater area for reduction than do lump ores and explain in part the superior reducibility of sinter.

In dealing with flotation concentrates the problem is how to make a layer of them uniformly permeable to air. If we can do that the rest is

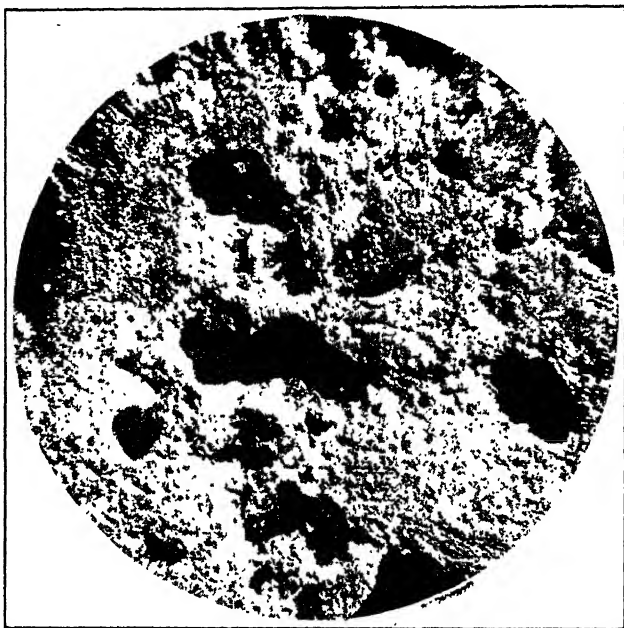


FIG. 1.—POLISHED SURFACE OF SINTER CAKE MADE FROM MISSOURI GALENA. $\times 50$.
(Reduced one-third; original magnification given.)

easy. With flotation concentrates that retain an appreciable amount of residual oil, the difficulties of treatment are increased. The ignition is slow and the sintering action sluggish, with sintering capacities reduced in consequence. The reason for this is not entirely clear, but it may be due to tighter contact between particles due to oil films thus reducing the area of the air passages. If the concentrates are heated to the point of steaming this oil is driven off and the subsequent behavior is much better.

An ideal condition is created by steaming the concentrates and rubbing them through a $\frac{1}{8}$ -in. screen forming a mass of pellets. These tend to take a partial "set," so that the pellets maintain their separate shape under ordinary handling, and a mass of them when fed to the

sintering machine gives an almost ideal charge. The individual pellets being loosely compacted are themselves relatively permeable and roast more speedily and thoroughly than solid ore particles of equal size.

These pellets can be produced by a rolling action, but the difficulty is that large balls form as well as small pellets, and the large nodules of flotation concentrates cause irregularities and poor desulfurization.

Various means are used for breaking up the wet lumps and obtaining a thoroughly uniform and well mixed charge, among which may be men-



FIG. 2.—PUG MILL OF SPECIAL DESIGN FOR FLOTATION CONCENTRATES.

tioned screening the charge and reworking the lumps; forcing the lumps through a screen; or passing the charge through a disintegrator such as a swing hammer or squirrel-cage breaker. However, the formation of lumps can be prevented to a large extent by using suitable apparatus for mixing the charge. The pug mill type of mixer is better adapted for flotation concentrates than the table type, as the churning action of the blades tends to break up lumps. However, with the ordinary pug mill, balls of concentrate may form and work along to the discharge end without being broken up sufficiently. To meet this difficulty, an improved form of pug mill has been devised; this consists of a slowly rotating drum

with an inner paddle shaft offset to one side and below the axis of the drum which revolves rapidly in the opposite direction to the drum. The rapid movement of the paddles thoroughly mixes and "fluffs" the charge, and lumps are continually carried by the rotation of the drum into contact with the paddles. Fig. 2 shows a pug mill of this type with some of the pelletized product.

Another method of sintering flotation concentrates, which is widely used, is to mix back with the charge a proportion of finished sinter crushed to minus $\frac{1}{2}$ or $\frac{1}{4}$ in., called "returns." These "returns" serve two purposes. Being low in sulfur themselves, they dilute the sulfur content of the mixture sufficiently so that a single sintering operation produces the desired low-sulfur sinter cake. The "returns" are comparatively coarse and therefore make the mixture loose and readily permeable to the air blast, while at the same time each piece of returned sinter becomes an inert nucleus coated with a thin layer of the sticky fine sulfide particles which are thus presented in an ideal way for rapid and efficient blast roasting. The result is that sintering takes place rapidly, and a much larger tonnage can be treated with a given equipment than would otherwise be possible. Instead of "returns," sometimes a small proportion of granulated slag is used for the same purpose. More care in preparation of the charge for sintering and the thorough mixing of the ingredients is required in this procedure than on straight double sintering, for if large lumps of concentrates are permitted to go into the feed, or if the coarse material is not evenly distributed through the fine, the sulfur in the sinter will be high.

SINTERING COPPER FLOTATION CONCENTRATES

To the copper metallurgist who has reverberatory furnaces available, smelting flotation concentrates does not present such great difficulties, since the reverberatory thrives on a fine feed, but where smelting is carried on in blast furnaces, or by the miner who wishes to reduce his freight cost and roasting charges and avoid the loss of material inevitable in handling these fine concentrates, sintering is frequently practiced with advantage. The usual method is that described, in which returns are mixed with the concentrates to make up the feed to the machine. Except that sulfur elimination is not carried so far (sufficient sulfur being left in the sinter to cover the copper and to give the desired matte fall in smelting) the operation is very much the same as in sintering lead flotation concentrates.

SMELTING ZINC FLOTATION CONCENTRATES

In the zinc field, even more trouble than with lead or copper was encountered in smelting flotation concentrates. They were usually somewhat lower in grade than the coarser concentrates, but entirely aside

from this the difficulties in roasting and retorting this exceedingly fine material were so great as frequently to find expression in an extra smelting charge against them and at times in refusal to buy them at all.

In the roasters the dry fine concentrates run almost like water, spreading out and causing high spillage and dusting losses. They form an impervious bed of which only the surface is actually roasting, so that furnace capacity is decreased and gas consumption and labor costs per ton are materially increased; besides, the roasted product is erratic and high in sulfur. They are equally objectionable in the retorts, for they work down through the charge to the bottom, forming a dense mass where distillation is slow and incomplete. In consequence the residues are high in zinc and recoveries are correspondingly low. Slagging of the retorts is greater, with decreased life of retorts and additional cost of renewals.

The zinc metallurgist desires a comparatively fine granular material which can be intimately mixed with the reducing agent to form the charge for the retorts. The Dwight & Lloyd sintering machine can be made to form such a product, which is ideally suited to the peculiar requirements of the retort process. The individual particles of flotation concentrate completely lose their identity in sintering and are transformed into a friable cake of cellular granules, which is readily crushed to the 3 to 4-mesh size desired for the retort charge. Sinter made from flotation concentrates cannot be told from that made from coarser materials, and distills equally well.

Zinc flotation concentrates are usually preroasted on hearth furnaces of the ordinary types, with conversion of the gases into acid. This roasting may be either the usual dead roast down to 2 to 3 per cent. sulfur or may be speeded up to leave 7 to 9 per cent. sulfur in the product. In the latter case, the capacity of the furnace is about double, the gas consumption about half that when dead roasting, therefore the cost of the operation is correspondingly lowered.

Concentrates preroasted to about 8 per cent. sulfur contain sufficient fuel for the sintering operation but if they have been dead roasted, 5 to 6 per cent. carbon must be added. This may be fine coke or coal, or crushed retort residues. A charge with carbon fuel sinters more rapidly than with sulfur fuel, possibly because the carbon particles are exposed and easily ignited whereas the sulfide particles left in preroasting are usually coated over with oxides and are less easily ignited and burned.

Greater care is necessary in preparing the charge for sintering than is the case with lead or copper ores, and Tri-State ores are somewhat more difficult than western ores to handle. This is due partly to the characteristics of the ores and partly to the peculiar requirements of the retort process. Tri-State ores particularly have a somewhat sandy character when roasted and tend to make a rather impervious bed unless the mois-

ture is held within close limits and mixing is carefully done. Coarse concentrates and sintered returns are often mixed in to make the charge more permeable. Then, for retorting, the sintered product must be minus 3 or 4 mesh size and as completely desulfurized as possible. Therefore a low-sulfur, friable, granular product must be made, which can be readily crushed without going to excessively fine particles.

The charge is made up from a series of automatic feeder bins, so that a uniform mixture can be maintained by continuously drawing low-sulfur preroast from one bin, high-sulfur from another, with returns, fuel, etc., as required. A pug mill usually is employed for mixing, and the mixture is fed to the pallets through a swinging spout distributor.

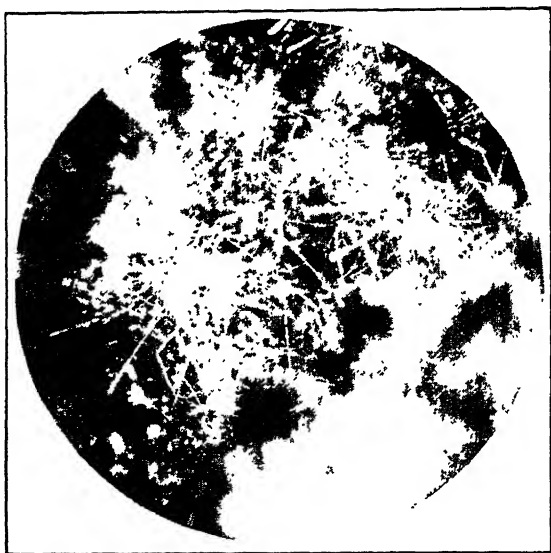


FIG. 3.—SINTER MADE FROM JOPLIN BLENDE PREROASTED TO 10 PER CENT. SULFUR.
× 25.

(Reduced one-third; original magnification given.)

Ignition usually requires a more prolonged or intense heat, for zinc sulfide is comparatively difficult to ignite and also the coating of oxide on the preroast makes ignition difficult. An ignition muffle 2 to 3 ft. long, gas fired, is commonly employed, gas usually being available at zinc smelters at lower cost than oil or other fuel. Western ores ignite more readily than Tri-State. In fact, as the lead and iron content of the charge increases, the difficulties in sintering decrease, except that the sinter is apt to be harder and somewhat more trouble to crush. Smooth rolls alone may suffice for crushing sinter from Tri-State ores, but these have to be preceded by spike rolls or corrugated rolls for that from western ores.

Once sintered, there is no difference between flotation concentrates and coarser materials. Both make a sinter equally low in sulfur—consistently below 0.5 per cent. sulfur in the case of sintering “dead roast”

with carbon and below 0.75 or 1 per cent. sulfur in sintering from 8 per cent. sulfur preroast. Physically the sinter is ideal for retorting. It crushes readily to granular condition and these granules themselves are cellular and porous, making distillation of the zinc easy and rapid. In the retort there is no tendency to segregate from the coal, as is the case with flotation concentrates; the residues do not slag badly, and the life of the retorts is prolonged. Less fuel and less "dead coal" are required, smelting may be carried on at a lower temperature, and recoveries have been found to be increased from 1.5 to 4 per cent.

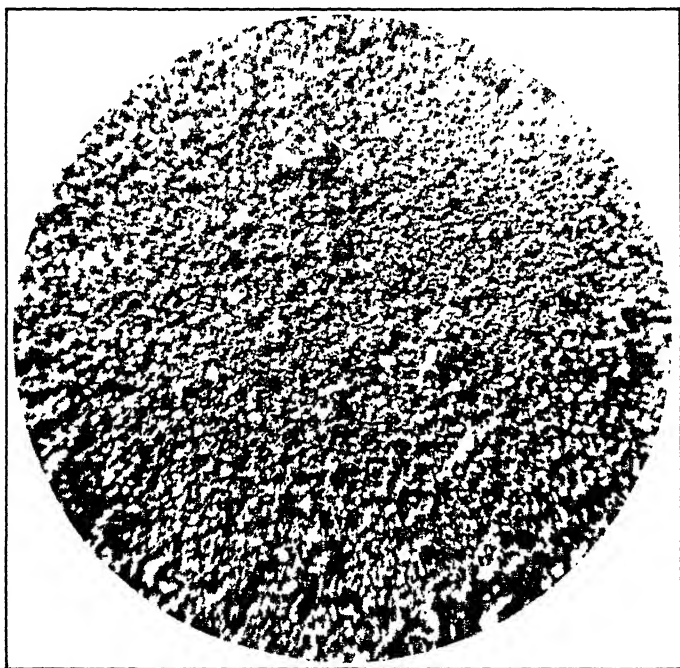


FIG. 4.—RAW WESTERN ZINC FLOTATION CONCENTRATES. $\times 50$.
(Reduced one-third; original magnification given.)

Contrary to what might be expected, there is no appreciable volatilization of zinc in sintering, when a nonvolatile coal fuel is used. No fume is visible coming from the stack, and baghouse tests have shown the dust loss to be too small to be measured. The small amount of dust found in the wind-box is less than 0.5 per cent. of the charge.

Sometimes a small amount of zinc oxide crystals may be seen in the cavities of the sinter. Apparently any zinc that may be volatilized is immediately deposited rather than carried away in the gas stream. That this is the case is indicated by the fact that the crystals may be found in the upper or lower portions of the cake, indiscriminately, and

that no appreciable amount of fume is collected in the bags. Fig. 3 shows a mass of these oxide crystals filling a cell hole in a sinter cake.

Fig. 4 shows raw western zinc flotation concentrates and Fig. 5 shows a polished surface of the sinter made from these concentrates after



FIG. 5.—POLISHED SURFACE OF SINTER MADE FROM CONCENTRATES SHOWN IN FIG. 4 AFTER PREROASTING. $\times 50$.

(Reduced one-third; original magnification given.)

preroasting. When it is considered that the entire area shown in Fig. 5 is actually about $\frac{1}{8}$ in. dia., it is at once apparent why this angular, porous material cannot sift through the coal in the retort but holds its place to be rapidly and completely distilled.

The Waelz Process

By R. HOFFMANN,* CLAUSTHAL, GERMANY

(New York Meeting, February, 1928)

THE Waelz process produces oxides of volatilizable metals from ores, metalliferous products and residues. The process was originally used for recovering zinc and lead, where tailings and residues containing those metals had accumulated at mines or smelters, but it is applicable for obtaining tin, arsenic or other volatilizable metals. It is useful when other methods of obtaining the metals from the ores or residues are too difficult or expensive or show too low a recovery.

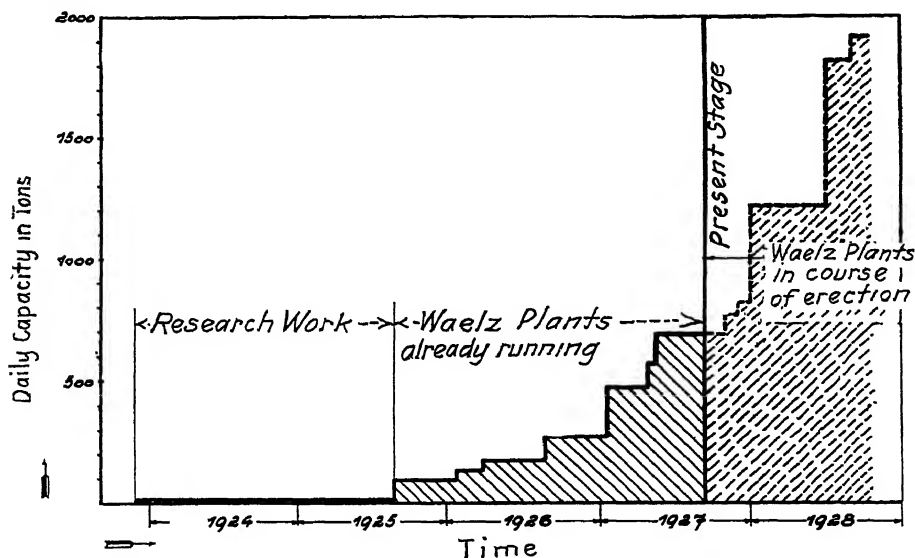


FIG.—1. DAILY CAPACITY OF WAELEZ PLANTS RUNNING OR BEING ERECTED.

The Waelz process was developed in 1923 for the treatment of large calamine dumps in Upper Silesia, which showed zinc contents of 8 to 10 per cent. and lead contents up to 3 per cent. The rapid development of the process, from the first trials up to the plants now working or in course of erection, is shown in Fig. 1. The first industrial Waelz plant was started in 1925. Within two years the capacity of the working plants reached 700 tons material per day of 24 hr. During the year 1928 the daily capacity of the plants now at work or in course of erection will rise

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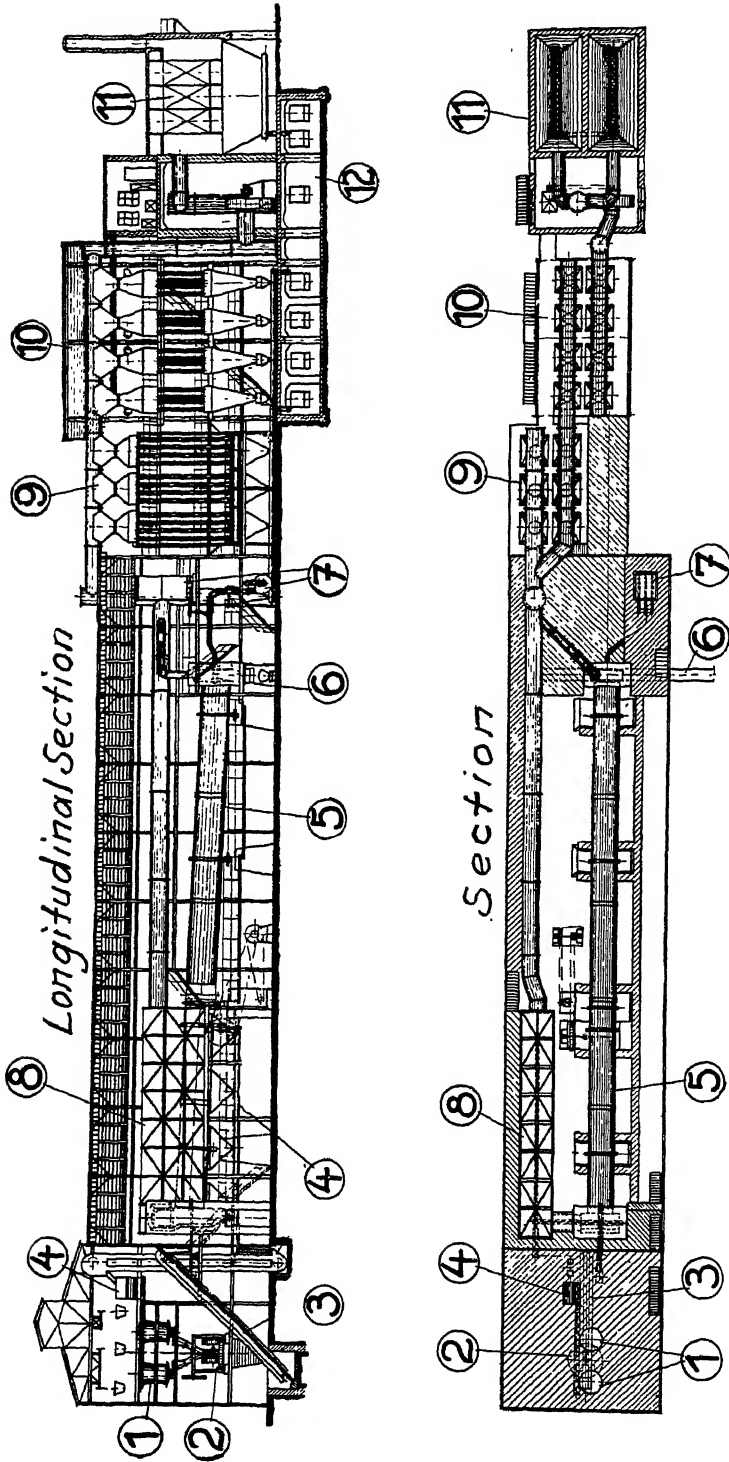


FIG. 2.—LONGITUDINAL SECTION AND CROSS-SECTION OF A WAELEZ PLANT.

- | | |
|------------------------|---|
| 1. Disk feeder | 7. Pulverizing-coal plant (unit system) |
| 2. Chilean mill | 8. Flue-dust chamber |
| 3. Bucket elevator | 9. Tubular cooler |
| 4. Return of flue dust | 10. Cottrell treater—pipes |
| 5. Rotary kiln | 11. Cottrell treater—plates |
| 6. Discharge door | 12. Oxide bagging room |

to about 1900 tons, which means that per annum about 50,000 to 60,000 tons of zinc and 10,000 to 12,000 tons of lead will be recovered in the shape of oxides from low-grade ores and residues by means of the Waelz process.

THE FLOW SHEET

The process usually is carried out as follows (see Fig. 2): The ore is mixed in hoppers (1) with fuel. The mixed charge passes through a crusher (2) and a conveyor (3) to the kiln feeder (1), which charges it



FIG. 3.—GENERAL VIEW OF A WAE LZ PLANT.

into a rotary kiln (5) adapted to the special conditions of the Waelz process. Cheap low-grade fuel such as coke breeze, anthracite duff or coal waste may be used. The ore (or tailings, residues) can be charged dry or as slimes.

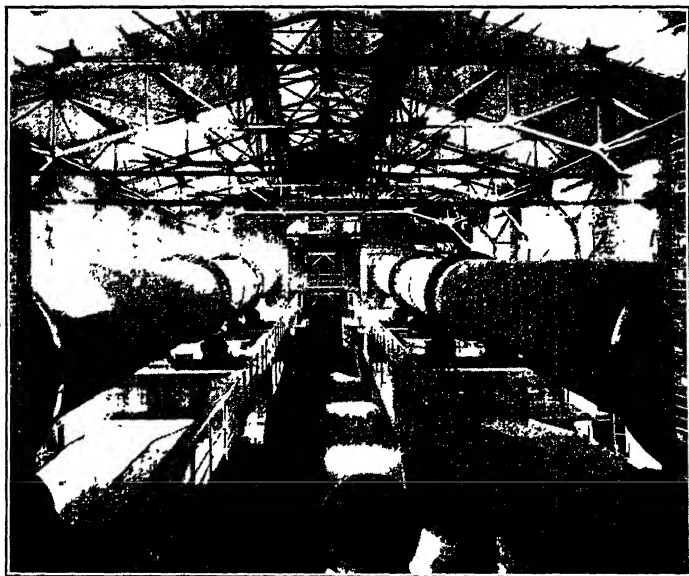


FIG. 4.—INSIDE OF A WAE LZ PLANT HAVING TWO KILNS.

In most cases the ore contains 10 to 15 per cent. moisture, but in some plants materials showing 30 to 40 per cent. of water are treated. The fuel consumption does not rise when moist material is being treated and the working in the kiln may be even better than when a material is too dry.

On the other hand, calcining of the carbonates or roasting of the sulfides is not necessary.

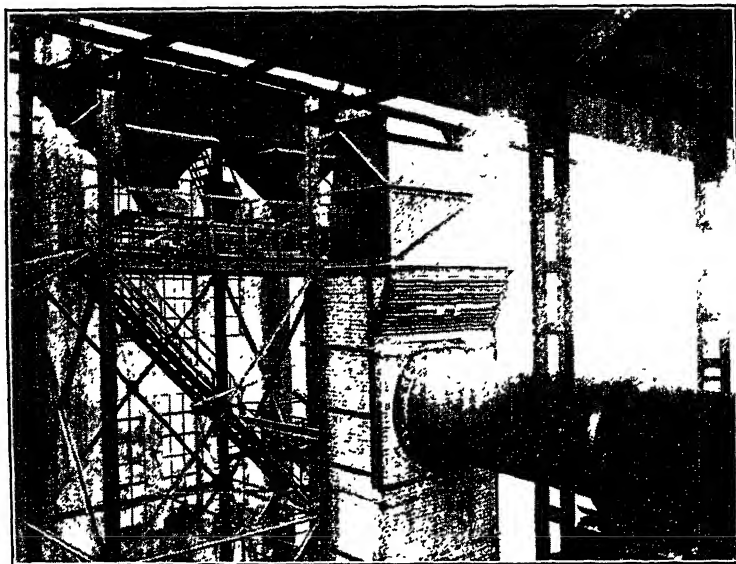


FIG. 5.—FEED END OF A WAELEZ PLANT, WITH HOPPERS FOR ORE AND COKE.

The residues leave the kiln at the lower end (6). As a rule the material takes about 2 hr. to pass through the kiln. The air required for the proc-

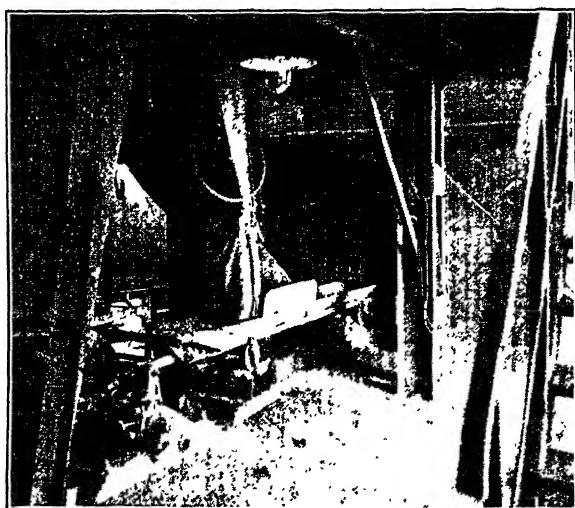


FIG. 6.—ORDINARY EQUIPMENT FOR FEEDING THE KILN—SHAKING FEEDER.

ess enters at the lower end where the residues leave the kiln, and flows, therefore, in the opposite direction to the charge. The gases leaving the

kiln pass through a dust chamber (8), then through a pipeline to the tubular cooler (9) and to the Cottrell treaters (10 and 11). The flue dust precipitated in the chamber is returned by an apparatus consisting of a

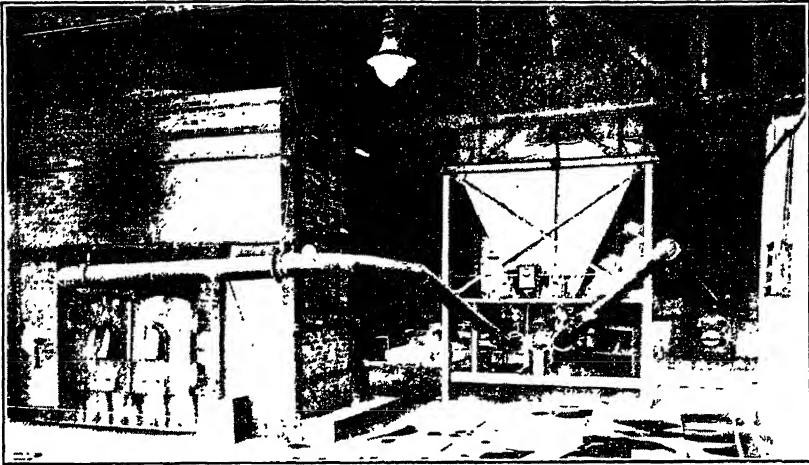


FIG. 7.—KILN WITH SERVICE PLATFORM AND BURNER FOR PULVERIZED COAL.

screw conveyor plus an elevator (4) to the hopper (1), to be remixed with the charge.

Under the Cottrell plant there is a series of hoppers (12) where the oxides from the cooler and the Cottrell are bagged.

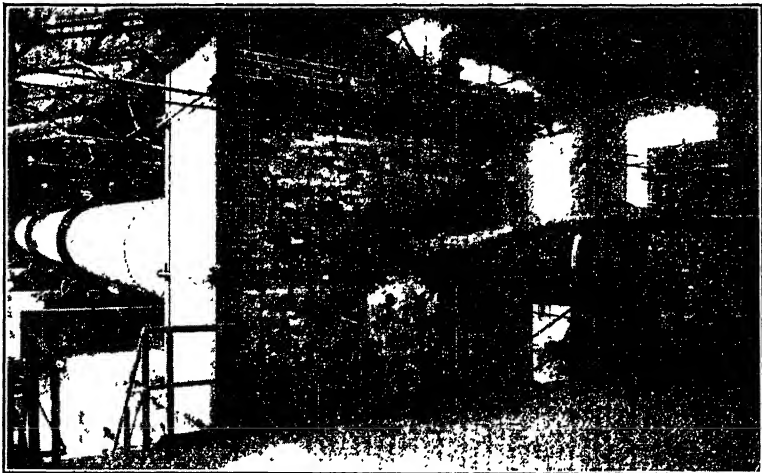


FIG. 8.—PLATFORM OF A WAELEZ PLANT—DISCHARGE AND FIRING END.

For starting the process and later on for regulating purposes an auxiliary heating mechanism (7) consisting of a burner for pulverized coal is arranged at the discharge end of the kiln. Oil can be used in some cases instead of the pulverized coal, if found to be more economical.⁷

Views of a Waelz plant and of parts of the apparatus are shown in Figs. 3 to 10.

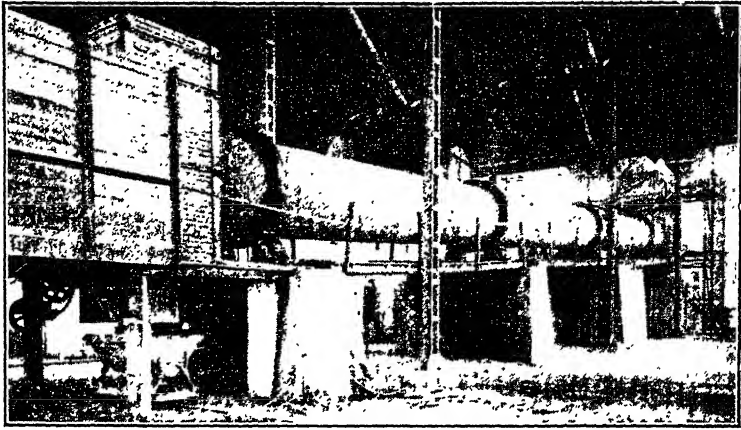


FIG. 9.—THE WAE LZ KILN, DISCHARGE AND FIRING END.

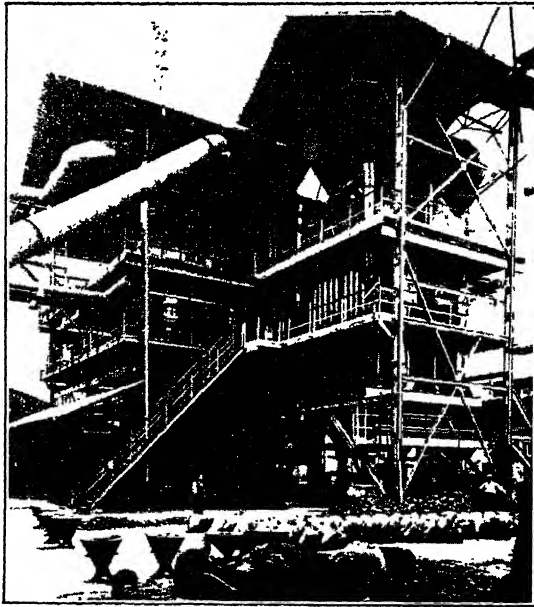


FIG. 10.—COTTRELL PLANT.

USE OF THE ROTARY KILN

Rotary kilns have seldom been used for metallurgical purposes, but they are in general use for the manufacture of cement. It goes without saying that the mode of working a cement kiln could not be applied unchanged to the running of a Waelz kiln, because the chemical conditions and the results are quite different. In the cement kiln, the flame is an

essential constituent of the process, the hot gases effecting the required physical and chemical modifications of the charge; also three different zones must be considered; the drying zone, the calcining zone and the sintering zone.

In the Waelz process, in a similar manner, the material fed into the kiln is dried and preheated to about 1000°C . at the upper end of the kiln. The heating is effected partly within—that is, on the surface of the charge—and partly indirectly, through the contact with the hot lining below the charge in each revolution.

The material preheated in this manner passes into the zone of volatilization where the temperature, of course, depends on the nature of the respective metal. For zinc, 1100° to 1200°C . is usually required, in order to attain a good volatilization. As the volatilization ceases when sintering commences, the recovery of metals from materials of a low melting point is, of course, much less than from materials with a high melting point. The melting point of the charge can be increased by adding suitable reagents such as limestone, but only if this be found economical.

CHEMICAL REACTIONS DURING VOLATILIZATION

The chemical reactions that take place during the period of volatilization are carried out in two stages; *viz.*, a reduction of the oxides takes place inside of the charge and the reducing gases and metal vapors escaping from the charge burn in the oxidizing kiln atmosphere. These two operations are described by the following two formulas:

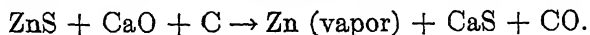
1. Inside the layer of material: $\text{ZnO} + \text{C} \rightarrow \text{Zn (vapor)} + \text{CO}$.
2. In the atmosphere of the kiln, above the material: $\text{Zn (vapor)} + \text{CO} + \text{O}_2 \rightarrow \text{ZnO} + \text{CO}_2$.

When the formulas are added one obtains as a summary of the several operations, inside and outside of the charge, the simple exothermic oxidation of the carbon used for reduction to CO_2 . It does not matter whether the several intermediate reactions proceed according to the formulas or whether still other intermediate reactions take place.

Those stages can well be observed when working. Over the whole extent of the zone of volatilization one can see continuously little green flames colored by the combustion of zinc. Towards the end of the volatilization zone, the zinc contents become less and in conformity the development of CO and zinc vapors cease. The flames become smaller and show less color, and pass over into a light yellow flame caused by direct combustion of the carbon on the surface of the charge. The occurrence of this flame is a good sign of thorough volatilization of the zinc.

The development of the metalliferous zinc vapors and of the CO on the surface of the material is of such a uniform extent that the oxidizing atmosphere in the kiln cannot touch the charge.

The Waelz process therefore can be used for the treatment of sulfide ores, without elimination of the sulfur by previous roasting. Under the reducing conditions inside of the charge of the Waelz kiln, the sulfide of zinc, for instance, is decomposed by oxides of nonvolatilized metals or earth alkalis, contained in the raw material or added to the charge, the sulfur remaining in the charge in the form of fireproof sulfides such as CaS, FeS, Cu₂S, etc. The volatilization of zinc then proceeds as follows:



The exclusion of the oxidizing gases from the charge is so complete that only insignificant traces of SO₂ are produced. Small quantities of sulfur which may be distilled by decomposition, for instance of FeS₂ into FeS + S are neutralized by forming sulfates with the oxides of zinc or lead. In all cases only traces under 0.01 per cent. of free SO₂ have therefore been found analytically.

The gases from a coal fire contain as a rule 0.1 per cent. of SO₂, therefore no notice need be taken of the SO₂ contents in Waelz plants.

HEAT USED IN WAE LZ PROCESS

Contrary to the cement process, the quantity of heat used in the Waelz kiln is not produced by a flame but almost exclusively by the combustion of the reduction gases and the metalliferous vapors. As the reduction of the zinc oxides in the charge needs the same quantity of heat as that liberated by the oxidation of the zinc vapors, the process shows only the following losses of heat:

1. The heat in the waste gas. The gases leave the zone of volatilization at a temperature of 1000° to 1100° C. In the preliminary heating zone, however, they give up the greater part of heat, the final waste gases only showing 300° to 600° C. In ordinary working the composition of the waste gas is: 0 to 1 per cent. O₂, 0 to 2 per cent. CO, 18 to 28 per cent. CO₂, which depends, of course, on the quantity of carbonates in the charge. As the contents of oxygen are very low, the quantity of gases is likewise comparatively small, which means that the loss of heat in comparison with other processes is small.

2. The heat taken up by the charge in the volatilization zone. A part of this heat is recovered for the purposes of the process by preheating the air entering the kiln.

3. The quantity of heat radiated through the kiln wall. For a good working, the heat produced in the entire zone of volatilization should cover these losses of heat specified under heads 1 to 3. Too great a development of heat at any place of the volatilization zone should, however, be avoided, because there is then a risk of accretions being formed. On the other hand, if too little heat is developed, it should be compensated

by a small flame. This flame can by no means increase the temperature in the entire zone of volatilization; it only preheats the air and thereby reduces the loss of heat. The temperature in the volatilization zone and the chief development of heat in that zone are regulated by the speed of the rotation of the kiln and by the amount of material fed.

COMPARISON OF WAELEZ KILN WITH CEMENT KILN

These essential differences in the working of the cement kilns and the Waelz kilns are shown in Fig. 11. The data for the first diagram dealing with the manufacture of cement are taken from the article by E. S. and E. Ernst, South Dakota Cement Plant, Rapid City, S. D.

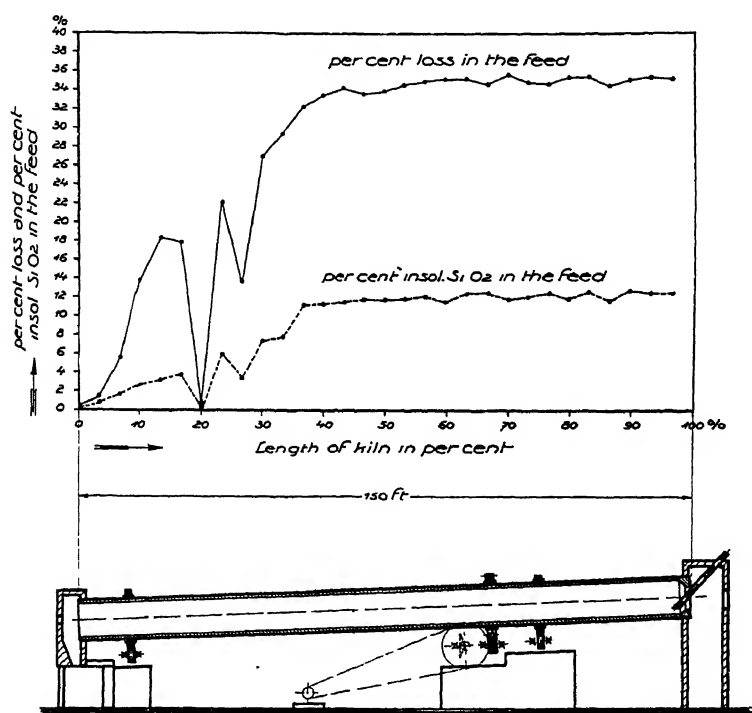


FIG. 11.—CHEMICAL PROCEDURE WHEN BURNING CEMENT.

The second diagram (Fig. 12) resulted from a Waelz test. The kiln was stopped for a moment during the normal working while the different samples were taken with spoons with long handles, at both ends of the kiln. This kind of sampling may give rise to some inaccuracy. By numerous tests carried out in this way it has, however, been proved that the figures found are sufficiently exact. During the trial shown in this diagram a zinc ore high in silica was treated. The kiln charge consisted of ore and coke breeze and contained 15.89 per cent. Zn.

In these diagrams, for the sake of better comparison, both kilns are drawn in such a way that the various lengths are given in percentages of the total length of the kilns. The diagram of the cement kiln shows that the preheating of the charge requires 60 per cent. of the total length. At 40 per cent. of this length begins the calcining and nearly at the same time the solution of the silica. On the other hand, in the Waelz process the heating of the charge only needs 20 per cent. of the total length of the kiln. By reducing the oxides of iron, eliminating the CO_2 as well as the water of

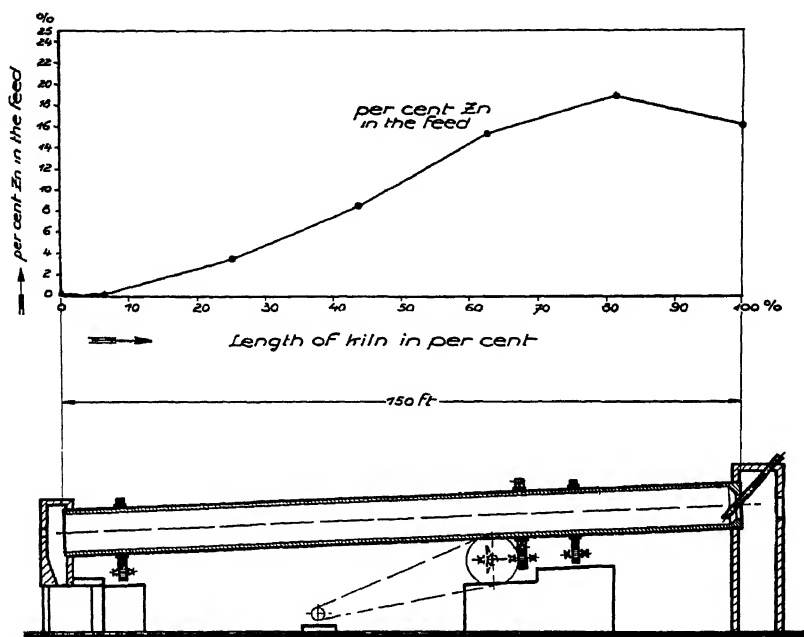


FIG. 12.—VOLATILIZATION OF ZINC IN THE WAELEZ PROCESS.

the hydrates, the zinc contents on the point marked "80 per cent." of the kiln length are higher than in the original charge. The material in this zone reaches the temperature required for reducing the zinc. The volatilization of zinc then takes place over the whole length of 82 to 6 per cent.; viz., in about 75 per cent. of the kiln length. The curve shows that the volatilization during all this time proceeds almost uniformly. The speed of this volatilization and the length of the reaction zone are determined by the speed of the kiln and the quantity of material charged, as already shown above.

CAPACITY OF PLANTS INCREASED

Development in industrial plants during the last two years has brought the following results:

The dimensions of the kilns have been gradually increased.

The trial plant at Fried. Krupp Grusonwerk A. G. of Magdeburg has been equipped with a kiln of a daily capacity of 10 to 15 tons.

The first industrial plant had a daily capacity of 40 to 50 tons, and the kilns installed later on of 80 to 120 tons. Some of the plants now being put up contain kilns of a daily capacity of 200 tons. One may assume that at the present stage of technology with kilns of 350 to 400 tons the maximum capacity will be reached.

COST OF THE WAEZL PROCESS

The cost of working Waelz kilns depends, of course, on local conditions with regard to expenses for fuel, wages and power. The consumption of waste fuel such as breeze, anthracite duff, etc., amounts to about 25 to 30 per cent. of the ore and the power needed to 20 to 30 kw.-hr. per ton, including transport to the plant.

Very little labor is required for Waelz plants, as charging and discharging can be arranged mechanically.

The working costs depend also on the size of the kiln, because fuel and power needed per ton of material decrease with larger kilns and the number of hands remain about the same as needed for small kilns. It may be presumed that under advantageous local conditions a material containing 4 to 5 per cent. of zinc may still be treated profitably, if the material comes from a dump or an existing plant, and if the commercial value of its zinc contents may be looked upon as being near zero.

This low economical limit shows the progress made by introducing the Waelz process for the treatment of materials not capable of being handled by the methods known hitherto. In one big plant an ore containing exclusively lead in the form of galena and cerussite is being treated. The ore is dressed wet mechanically, concentrated up to 13 per cent. Pb and then subjected to the Waelz process. The residues contain less than 0.5 per cent. Pb and the oxides from the Cottrell collector about 78 per cent. Pb containing a mixture of lead oxide and lead sulfate.

PURITY OF OXIDES OBTAINED

To some degree the volatilization of metals depends, as already stated, on the melting point of the charge. In cases of a low melting point the volatilization may drop to 85 per cent., while in general it amounts to 90 per cent. and more.

The purity of the zinc oxides depends principally on the speed of the gas leaving the kiln in the flue-dust chamber. In one plant treating calamine of 10 per cent. Zn + Pb an oxide containing 68 to 75 per cent. of Zn + Pb is obtained. The flue dust, which returns automatically to the charge, amounts to 2 to 3 per cent. of the ore and contains 30 to 45 per cent. Zn + Pb.

The inclination of the charge to form accretions varies considerably with the different raw materials. It happens most frequently with charges having components with widely different melting points. These accretions are caused in all cases by local excess of heat attributable to lack of uniform charging. If the working of the plant is well regulated, no difficulties from accretions should be expected.

Sometimes by adding reagents such as common salt to materials containing sulfur, various reactions of an accessory nature were created, in order to increase the formation of volatile compounds of metals.

Finally it has been shown that in many cases the metals contained in the material can be recovered in several products. For instance, zinc and lead may be recovered separately by volatilizing the lead in the first stage of working and following this operation by a process for volatilizing the zinc. If the lead contents of the ore are low, it will be more economical to volatilize $Pb + Zn$ together. The recovered oxides, containing perhaps 5 to 10 per cent. Pb , can then be retreated in order to recover a lead product. The zinc oxide is then not volatilized, but discharged in a sintered state from the kiln. This agglomerated zinc oxide is considerably better than the nonagglomerated oxide for the manufacture of spelter in muffles.

DISCUSSION

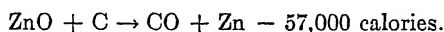
A. L. QUENEAU, New York, N. Y. (written discussion).—The Waelz process is essentially a fire method of concentration. This is well shown in the case cited in the paper of a plant in Silesia where the treatment of old tailings (zinc plus lead 8 to 13 per cent.) gave a fume with 65 to 75 per cent. zinc plus lead. In my own experience, also in Silesia, from a zinc furnace residue assaying 4.73 per cent. zinc and 2.14 per cent. lead, I have obtained a mixed fume carrying 54.8 per cent. zinc and 18.8 per cent. lead. The Waelz process allows, further, the separation of the various metals contained in any given ore, according to the order of volatility of their compounds. Thus lead, zinc, arsenic, cadmium, etc., may be separated from iron, manganese, copper, nickel, etc. Silver goes in part with the fume, the balance remaining in the kiln residues; gold will usually stay in the residues, especially if copper is present.

The process is carried out in a rotary kiln of the same general type as that used in the Portland cement industry. The similarity between the cement process and the Waelz process may be emphasized. In both processes it is essential that sintering and not fusion should occur, and the mass under treatment should remain at all times in a practically "dry" state. The volatilization of zinc and lead is hindered when a pasty condition is reached, the operation of the kiln becomes difficult with the semi-fusion or fusion of the charge, and wall accretions in the form of rings will then take place.

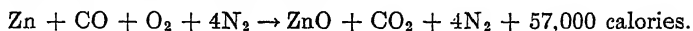
In the Waelz process, the required fuel for heating and for reduction is usually added to and mixed with the charge. This is in contradistinction to cement practice, where firing takes place at the discharge end of the kiln, the fuel being pulverized coal, natural gas or oil.

The charge in a Waelz kiln usually forms a long, continuous cylinder, called a "sausage" in Germany. In the zone of volatilization the conditions within the "sausage" are very like those prevailing within an ordinary zinc retort. There is

present a large excess of glowing carbon, finely disseminated and in intimate contact with the particles of zinc and lead compounds. Reduction takes place with the formation of metallic zinc in the form of vapor and carbon monoxide, according to the chemical reaction



The liberated gases find their way to the surface of the cylinder, where in contact with an oxidizing atmosphere, air being admitted at the discharge end of the kiln, they are reoxidized according to the reaction



There is thus a balance between the endothermic and the exothermic reactions. Beyond the fixed carbon required for the above reaction, fuel is also needed to maintain the kiln at the reacting temperature. Heat losses occur as follows:

1. Radiation.
2. Calories carried away by the products of combustion with their fume burden.
3. Dehydration and calcination of the charge.
4. Calories carried away by the final residues.

From the above, it would appear that the fuel requirements beyond those necessary for reduction, for a given tonnage, should follow closely those of the cement industry, 18 per cent of fuel or less, varying with the size of the kilns, process, etc., reckoned on the clinker production. The fuel requirements in the Waelz process usually vary between 25 and 30 per cent. of the weight of the kiln feed.

Again, as in the cement industry, the Waelz kiln requires a close chemical control of the charge, care being taken that the final residues are of such composition that they are infusible at the temperature of the discharge zone. As in zinc retorting practice, the presence of an excess of carbon in the final residues will be of material assistance in preventing the coalescing of fused particles.

As pointed out by Professor Hoffmann, uniformity of operating conditions, such as rate of feed, physical and chemical character of the charge, temperature, etc., makes for ease of operation and freedom from wall accretions and rings. Prevention is better than cure; it is easier to avoid rings than to remove them.

The zinc and lead fume of the same fineness as that of zinc oxide manufacture is carried away from the kiln in the products of combustion and is collected either in bag rooms or in appropriate Cottrell electric precipitators, tube or plate type, or a combination of both.

The draft requirements are met by exhaust fans placed either at the tail end of the system in the case of Cottrell, or ahead of the bag room.

The paper presented by Professor Hoffmann is timely, as it gives us a picture of the present-day status of the Waelz process.

A. L. QUENEAU (oral discussion).—I wish further to emphasize the very close analogy between the Waelz kiln and the cement kiln. The trend of the cement practice has been to compound the charge very fully, thus we are passing very rapidly from the dry process of cement manufacture to the wet process, and I think one of the main advantages of the wet process is that there is very intimate contact between the particles of the various materials forming the charge. In the same way, in the Waelz plant, the charge should be very thoroughly mixed and the crushing should be pushed quite far, at least through 10 mesh. The recovery is especially good with lead; it can be said that in the case of lead, at least with zinc furnace residues, the recovery of lead is total. In fact, we gather more lead than the sampling and analyzing of the furnace charge will show by the ordinary methods of analysis. The recovery for zinc can be depended on to be 90 per cent. and over. With German furnace residues which, as stated, were a little over 4 per cent., the recovery was over 88 per

cent. Of course, it is not expected that a pigment grade of oxide will be made. The oxide obtained, for lack of a better name, could be called a metallurgical oxide; that is, an oxide that will be stock material for further treatment, either for the production of slab zinc or as a lithopone base, or for other uses.

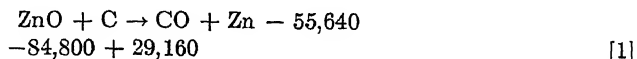
The separation of the lead, and maybe silver, contained in this oxide, probably would have to take place along the line indicated by Professor Hoffmann, or perhaps better, in a hydrometallurgical way.

A. SCHMIDT, Los Angeles, Calif.—There is no question but that the development of the Waelz process constitutes a valuable forward step in the treatment of certain ores, as well as zinc residues, but I would like to call attention to one point of historical interest, so that credit can be given where credit is due, for early pioneering work. In Dr. Hoffmann's paper, much emphasis is laid upon the difference between the Waelz process and the portland cement process, but it seems to me that we should not forget the early pioneering work done by Pohle and Croasdale in the 90's, in their endeavor to accomplish the volatilization of metals from ore by the chloridizing-volatilization process. They used a rotary kiln and endeavored to collect the fume in a bag house. Many years later, the Cottrell process of electrical precipitation was also used in connection with experimental work following up the Pohle and Croasdale investigations on the chloridizing-volatilization process. These investigators all encountered the slagging difficulty of which the importance is emphasized in Dr. Hoffmann's paper, and no large-scale commercial installations were built. Apparently the essential features of the present Waelz process, differentiating it from the early work referred to, lies in the fact that a reducing agent is added to the charge in the kiln. This brings about different chemical reactions while the presence of the coke breeze and the absence of salt apparently eliminate the fusion difficulties to a large extent. After all, however, the general idea of volatilizing metals by treating ore in a rotary kiln and collecting the resulting fumes for the recovery of the metal values, can be carried back more than 30 years.

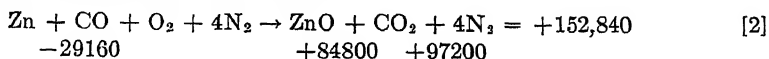
A. L. QUENEAU.—I might emphasize the point raised by Mr. Schmidt as follows: Really, in the Waelz process, we have a magnified zinc retort. We have no special reactions taking place that do not take place in the ordinary zinc retort. In the bottom of the large rotary kiln there is a bolster or a sausage of charge that may be 2 ft. thick. The kiln revolves very slowly, perhaps at the rate of 1 r.p.m. or $\frac{3}{4}$ r.p.m., and in this sausage, in this bolster, revolving slowly, unbroken and continuous, we have exactly the same conditions that prevail within a zinc retort; the reactions that take place are the reactions between the carbon and the zinc oxide, spinels, etc. We see the gases resulting from the reactions, forcing their way through the loose walls of the bolster and burning in contact with the oxidizing atmosphere. No new chemical reactions are involved: it is purely the old zinc process.

However, as we know, we cannot collect metallic zinc in an oxidizing atmosphere.

The equations given on page 549 should be corrected and completed as follows:



The gases are oxidized as per the equation



The final thermal balance of the equations 1 and 2 is a heat liberation of 97,200 calories.

Equation 1 takes place only at a temperature of 1100° C. and over; temperature at which the kiln lining and the charge are raised and maintained.

The fuel requirements of the Waelz kiln are usually met by the combustion of 25 to 30 per cent. of the weight of the charge. The carbon required for the reduction

of the zinc oxide, equation 1, must be supplied as fixed carbon, preferably in the form of coke breeze; the remainder of the fuel may be soft coal, oil, tar, gas, etc.

It may be added that zinc and heat are inimical; i.e., under reducing conditions, the higher the temperature of operations the lower will be the zinc contents of the final residues; the temperature of economical operations being limited only by the resistance of the refractory lining of the furnace.¹

W. G. WARING, Webb City, Mo. (written discussion).—The Waelz process is founded upon correct principles applicable to the reduction of zinc in contact with carbon, or of zinc sulfide in contact with lime and carbon, at a cherry red heat (1150° to 1200° C.), provided that the materials do not fuse at that temperature; for, as Professor Hoffmann says, "the volatilization of zinc ceases when sintering commences."

There is abundant proof that zinc cannot be volatilized from molten slags to any considerable extent, even by bessemerizing with a hot-air blast, at any temperature short of the boiling point temperature of zinc oxide (1400° C.). At this temperature, however, it can be completely volatilized from slags, or from any other material. It is possible, I believe, that many zinc sulfide ores could be fumed successfully if fed in a fine powder into a Stetefeldt shaft furnace of sufficient height, provided with an abundant supply of oxygen, without the aid of any carbonaceous fuel at all, after the furnace is preheated to 1400° or over, if such a furnace could be constructed to withstand the tremendous heat of reaction.²

Mr. QuenEAU says that the Waelz process allows, further, the separation of the various metals contained in any given ore, according to the volatility of their compounds. It is with regard to this point that I wish to call attention to the important difference between zinciferous fume produced by the low-temperature "reduction-volatilization-oxidation" process and that produced by simple high-temperature "oxidation-volatilization," without prior reduction or other preparation.

When complex zinciferous ores, especially those containing fluorine, arsenic, selenium, tellurium, molybdenum, or any one of these elements, with silica, are fumed by low-temperature reduction-volatilization-oxidation, the fume product does not consist of simple oxides, or oxysulfates, but is a mixture of many complex combinations, as shown by the fact that the zinc contained in the fume is more or less insoluble in such usual solvents as ammonium carbonate, or dilute sulfuric acid, while on the other hand that produced from the same complex ores by high-temperature oxidation-volatilization alone yields its zinc content completely to such solvents.

The reason for this difference may be inferred from a comparison of the volatilization points of the elements and their oxides as shown in Table 1, it being assumed that the volatilization points as determined are nearly identical with the temperatures of condensation, which up to now have not been well investigated.

The reduction of zinc oxide to elemental zinc by heating with carbon requires a temperature approaching 1100°, in practice. Antimony and bismuth oxides reduce at somewhat lower temperatures, while cadmium, tin and lead are reducible at 800° or less. It will be noticed that Zn is volatile at 470° below the volatilization temperature of its oxide; lead at 52° below the volatilization temperature of PbO, while all the other volatile metallic elements have higher boiling points than those of their

¹ This follows from Le Chatelier's theorem: "Any change in the factors of equilibrium from outside is followed by a reverse change within the system." The reduction of zinc oxide absorbing heat, an increase of temperature will be followed by an advance in zinc reduction.

² W. G. Waring: Zinc Compounds at High Temperatures. *Min. & Met.* (1925) 6, 610.

oxides; also, that the sulfides of Zn, Cd, Pb, Sb and As are all volatile at temperatures very much below that at which gaseous ZnO condenses into fume. It follows,

therefore, that if all these volatile elements or their sulfides (such as are capable of sublimation without decomposition) are liberated in the gaseous (vapor) phase at a temperature of, say, 1100°, in a reducing atmosphere, and these vapors are subsequently oxidized by contact with atmospheric oxygen while cooling, a system is presented in which a very large number of eutectic combinations can take place, with zinc as component of possibly the greatest number of them.

On the other hand, if the same elements are liberated *ab initio* as oxide vapors in a completely oxidized atmosphere, at or above the temperature at which zinc oxide sublimes and the vaporized oxides are cooled gradually, the vapors will condense selectively in the order shown in the first line of figures in Table 1, the ZnO first, as chemically pure acicular crystals, and the other oxides and oxysulfates as pure white fume particles, including zinc oxide, of course, but no insoluble complexes. That this is the case has been proved by a large number of actual furnace and leaching tests, conducted upon a semicommercial scale, thus opening a previously unexplored field in non-ferrous metallurgy, of greater importance, probably, than selective flotation, because its application rests on a simple scientific basis and its products are separable into definite homogeneous chemical groups instead of indefinite mechanical mixtures of slimed minerals.

R. HOFFMANN (written discussion*).—I agree with Mr. Quenau that the preliminary work offered by the cement industry in connection with the apparatus for the rotary kiln method was exceptionally valuable for the rapid introduction of the Waelz method in practice. The Grusonwerke especially, which had had extensive experience in the construction of cement kilns, was able to apply advantageously the knowledge obtained there, in the construction of Waelz installations. However, the similarity of the two methods is only external; the execution of the Waelz process is vastly different because of the different thermal processes in the kiln. The details of the processes are evident from the dezinzing diagrams in the paper.

As regards the further statements of Mr. Quenau, they give me no reason for taking a stand on one side or the other. A yield of more than 100 per cent. lead is explained by experiences obtained

TABLE 1.—Volatilization Temperatures at Atmospheric Pressure

Oxides..... Volatilization tem- peratures.....	ZnO 1400° C.	SnO ₂ 1230° C.	Sb ₂ O ₃ 1060° C.	PbO, PbSO ₄ 952° C.	Tl ₂ O ₃ 875° C.	MoO ₃ 760° C.	TeO ₂ 700° C.	CaO, CaSO ₄ 700° C.	SiO ₂ 390° C.	BaO ₂ 357° C.	As ₂ O ₃ 150° C.	SbF ₃ -65° C.	SO ₂ -10° C.
Sulfides..... Volatilization tem- peratures.....	ZnS 1180°		Sb ₂ S ₃ 540°	PbS 1085°				CdS 980°			As ₂ S ₃ 565°		
Elements..... Volatilization tem- peratures..... Melting points....	Zn 930° 419.4°	Sn 2270° 231.9°	Sb 1330° 630°	Pb 900° 327.4°	Tl 1280° 302°	Mo 3617° 2250°	Te 1390° 152°	Cd 776° 320.9°	Se 690° 180°	Bi 1490° 271°	As 360° (Sublim.)	P -187° (Gas)	S 444.6° 119.25°

* Translated from the German.

in other plants. It evidently must be explained without more ado by the inaccuracies in lead analysis, as it is obviously not easy to determine lead content of from 1 to 2 per cent. so exactly that the lead balances in the charge and product.

The statements by Mr. Schmidt, which are essentially historical, were refuted by my paper on the Waelz method, which was read before the Gesellschaft Deutscher Metallhütten und Bergleute in Berlin on May 3, 1928, properly supplemented (see *Metall und Erz*, 1928).

The statements of Mr. Waring give rise to the question as to whether the Stetefeld furnace is at all suitable for volatilization processes. There are considerable errors in the table of volatilization temperatures given by Mr. Waring. This table contradicts not only all previous scientific investigations, but also all direct practical observations when it asserts that zinc oxide can be volatilized at 1400° C., and zinc sulfide at 1180° C. A process that depends on the direct volatilization of the oxide has been developed by the Grusonwerke (Friedrich Krupp, Grusonwerk A.-G., Madgeburg-Buckau) in the so-called sintering method for removing zinc and lead from flue ash. Any extensive application of this method to zinc must fail from the beginning, because at the temperatures given by Mr. Waring the volatilization of zinc compounds has not yet started.

As regards the other data given by Mr. Waring, I would like to advance the figures given by Professor Kohlmeier in his address before the Gesellschaft Deutscher Metallhütten und Bergleute on Sept. 8, 1928, with the reservation that an explanation of the question in all its details is not yet possible but that it is absolutely necessary, before taking a further stand on the question, to view the detailed results of the experiments quoted by Mr. Waring. Professor Kohlmeier³ reaches the results shown in Table 2.

TABLE 2.—Data from Kohlmeier's Paper

Metal			Metal Oxide				Metal Sulfide				
	Melt- ing Point	Boil- ing Point		Melt- ing Point	Volatilization Begins	Boil- ing Point		Melt- ing Point	Volatil- ization Begins	Volatil- ization in Prog- ress	Boil- ing Point
Sn	232	2270	SnO ₂		~1500	?	SnS	870	?	330	?
Cd	321	766	CdO		900	1390	CdS		?	980	?
Pb	327	1555	PbO	883	800	1475	PbS	1112	?	950	?
Zn	419	906	ZnO		1300	>2000	ZnS		?	1182	?
Cu	1083	2305	{ Cu ₂ O CuO }	1230	Decomposition with evolution of oxygen	?	Cu ₂ S	1130	?		?
Fe	1530	3200	{ Fe ₂ O ₃ Fe ₃ O ₄ }	1565 1527		?	FeS	1180	?		?

³ Kohlmeier: Zur Flüchtigkeit von Schwermetallverbindungen in hüttenmännischen Prozessen. *Metall und Erz*. (1928) 25, 425.

Twinning in Metals

By C. H. MATHEWSON,* NEW HAVEN, CONN.

(Institute of Metals Division Lecture,† 1928)

MICROSCOPIC metallography has been exploited quite well enough to bring about a very general understanding that the typical metal or alloy is composed of minute crystalline particles blended into a coherent microstructural mosaic. One does not have to be a specialist in metallography to realize that the properties of such an aggregate are essentially a summation in appropriate form of individual effects derived from the shape, size, placement or orientation and cohesive characteristics of the component particles.

It is clearly of great importance to consider the various forms of discontinuity which may occur at the boundaries between these crystalline particles. When a number of particles each possessing the same orderly arrangement of atoms are brought together into a close-fitting system of purely haphazard contacts, not unlike a handful of snowflakes compacted into a snowball, the particles are said to possess random orientation. There are no generally accepted views concerning the arrangement of the atoms or the constitutional reaction between atoms where one crystal meets another, but these contact regions are characterized by strength rather than weakness and it is customary to require the presence of many rather than few boundaries in preparing metal for useful service.

Under certain circumstances all of the particles in a metal may be nearly alike in orientation. Other general tendencies of orientation may be associated with particular forms of mechanical and thermal treatment.

In contrast with these cases of fortuitous or statistical diversity of orientation, we often find adjacent particles united along a plane which possesses a grouping of atoms belonging equally well to both structures. This, of course, determines a fixed relationship between the two orientations and it is always possible to derive one from the other by some form of rotation or reflection prescribed by the symmetry of the crystal structure under consideration. Particles united in this manner are known as twin crystals although the term refers to the form of association rather than the number of individuals concerned.

* Professor of Metallurgy, Yale University.

† New York Meeting, February, 1928.

Twinning, in common with other features pertaining to the crystalline habit of minerals, is of value in identifying them and has been thus far considered chiefly from that point of view. In the field of metallurgy it is important as a form of structure alteration which appears to be vitally concerned in many metallurgical processes. In approaching this subject I have first sought to discover the extent to which twinning has been observed in the general list of metals.

TWINNING IN THE COMMON METALS

All of the metals which come frequently into the hands of every observer and may therefore be termed common metals—namely, copper, silver, gold, iron, aluminum, zinc, lead, tin, nickel, and platinum—are quite definitely known to occur in the form of twinned crystals, with the possible exception of aluminum, after an appropriate life history in which stress and heat treatment are chiefly instrumental.

In some of these cases, twinning was recognized in native metals or in special preparations, *e. g.*, isolated crystals, long before the day of wholesale examination of metal structures by the methods of the metallographic laboratory. Thus, crystals of native gold and copper twinned along an octahedral plane are described by Dana¹ and the large handbooks² tell us that twinning of the same form has been observed in natural crystals of the other face-centered cubic metals, silver, lead, platinum (and iridium).

The ordinary microstructures developed in metals by polishing and etching do not lend themselves readily to the quantitative interpretation of crystallographic characteristics, aside from the size of the crystalline grains, but they do permit instant recognition of the twin relationship in so far as this is revealed by the customary rectilinear trace of the twinning plane in the surface under observation and the uniform coloration of polysynthetic twin lamellae, which often occur in great profusion. These structural features give character and individuality—in fact, an impressionistic cubist appearance quite evident in Fig. 1—to the familiar photomicrographs of hot-worked or cold-worked and annealed copper, silver, gold, and their alpha solutions.

Copper and Alpha Brass

Although it is generally assumed that recrystallized copper and alpha brass twin along octahedral planes, Tammann and Meyer³ report twin-

¹ E. S. Dana: Textbook of Mineralogy, 3d Ed., 1922, 173, 350 and 353. John Wiley & Sons, Inc., New York.

² E. S. Dana: System of Mineralogy, 6th Ed. 1892. John Wiley & Sons, Inc., New York.

P. H. Groth: Chemische Kristallographie, Part I, 1906. W. Engelmann, Leipzig.

³ G. Tammann and H. H. Meyer: Die Aenderung der Kristallitenorientierung bei der Rekristallisation von Kupfer. *Ztsch. für Metall.* (1926) 18, 176.

ning along trapezohedral planes of form $\{211\}$ in their samples of recrystallized copper, some of which were prepared by mildly cold-working sections taken parallel to the cooling surface of a casting and then annealing for a period of one hour at a temperature of 750° to 800° C. Moreover, quite recently C. F. Elam,⁴ in an X-ray examination of what appeared to be well-marked twin bands in a sample of native copper, could find no important crystal plane or direction common to both parts of the structure.

No crystallographic measurements sustaining the first assumption were discovered, although it can scarcely be doubted that others in common with observers in the Hammond Laboratory have found slip



FIG. 1.—TWINS IN ALPHA BRASS.

bands, universally held to follow the octahedral planes, parallel and otherwise appropriately related to the twin bands in strained samples of annealed copper or alpha brass. In order to test this assumption further, A. J. Phillips⁵ prepared samples of coarse-grained copper and 70:30 brass cut and polished on two planes intersecting at right angles so as to permit measurement of the angles between two, three, or even four hypothetical twinning planes in a given crystal and the edge in each surface plane. The poles of these planes were then located in a stereographic projection and the distances between planes were measured. The two copper crystals and eight brass crystals investigated all gave measurements approximating to the nearest degree of arc the correct angles, 70° , $31'$, $44''$ or 109° , $28'$, $16''$ between octahedral twinning planes, and

⁴ C. F. Elam: Banded Structure in Aluminum and Copper. *Nature* (1927) **120**, 259.

⁵ A. J. Phillips: Twinning in Copper and Brass. *Proc. Inst. Metals Div., A. I. M. E.* (1928) 429.

furthermore a similar analysis of internal twin bands in both parts of a twinned copper or brass crystal left no doubt that the twinning axis was perpendicular to the observed composition plane.

This work proves that twinning in recrystallized copper or alpha brass sometimes if not always occurs parallel to an octahedral plane around an axis normal to this plane.

Steel and Iron

It is well known that the austenitic steels, which may be considered to represent gamma iron, yield twinned structures which are scarcely to be distinguished from the familiar copper and brass structures. This twinning has been observed in plain carbon steel in the form of a record left after etching at a suitably elevated temperature,⁶ although it is not possible to hold the metal wholly in the gamma condition on cooling. Osmond and Cartaud⁷ affirm that "the octahedral faces are at the same time planes of translation, planes of twinning, and planes of junction" in gamma iron and this applies to twins produced by simple deformation as well as to those produced by annealing after deformation.

Twinning in ferrite, or body-centered cubic iron, has been precisely described in a recent paper by McKeehan⁸ who, by analyzing the specular reflection from bright cube faces left by etching in dilute nitric acid, finds the twin crystal united along planes of form $\{211\}$ in such manner that a common orientation would result by rotating one part through 60° around an axis of form $[111]$; viz., one of the axes of threefold symmetry. S. Tamura⁹ within the past year has reported an occurrence of occasional bands in the ferrite grains of wrought iron and mild steel under the name "pseudo twins," not intended to define them specifically as pseudomorphs of earlier bands in austenite, but to emphasize his impression that metals which crystallize in the body-centered cubic lattice seldom, if ever, form annealing twins.

The twins observed in Armco iron wire by McKeehan were produced probably as a result of incidental mechanical disturbance during the process of growing a crystal of alpha iron by causing it to feed on a reserve of transforming gamma iron supplied continuously by uniform progression of the wire through a suitably heated zone. Although the wire was cold-worked to begin with, the temperature conditions set up in the process would appear to require that these twins be regarded as

⁶ M. A. Baykoff: Sur la Structure des Aciers aux Températures Élevées. *Rev. de Met.* (1909) **6**, 829.

⁷ F. Osmond and G. Cartaud: The Crystallography of Iron. *Jnl. Iron and Steel Inst.* (1906) **71**, 444.

⁸ L. W. McKeehan: Twinning in Ferrite. *Proc. Inst. Metals Div., A. I. M. E.* (1928) 453.

⁹ S. Tamura: Notes on Pseudo-Twinning in Ferrite. *Jnl. Iron and Steel Inst.* (1927) **115**, 747.

annealing twins rather than mechanical or deformational twins, following the form of classification so generally adopted by participants in this field of research. I question the inherent value of this distinction and look forward to the time when sufficient evidence is available to prove the origin of all twin structures. At present it may be confidently asserted that twinning, quite in common with recrystallization, never occurs as a result of annealing metals which have passed tranquilly from the liquid to the solid state but is induced by strain and that recrystallization and twinning are mutually reactive at some stage of the combined deformational and annealing process. Thus all twins may be of mechanical origin and outstanding differences due to their original dimensions and their subsequent behavior on annealing.

Neumann Bands

About twenty-five years ago O. Mügge¹⁰ made a careful crystallographic study of the characteristic lamellae, commonly known as Neumann bands, which ordinarily appear when iron is deformed by sudden impact. An earlier observation by Linck¹¹ that these lamellae followed planes of form $\{211\}$ was confirmed by Mügge who observed the directions of the lamellae on cubic cleavage planes and measured the angles of contact between the intersecting surfaces. A complete interpretation of the results was offered, according to which twinning along the plane (112) brought about a transformation of (010), (100) or (001) planes into ($\bar{1}\bar{1}2$), ($\bar{1}12$) or (110) planes, respectively.

Some twenty years later, Mügge¹² pointed out that on the basis of the twinning characteristics previously established, alpha iron would have to crystallize in the body-centered form of the cubic lattice, as has been amply demonstrated by the modern X-ray methods.

In spite of the definite outcome of this pioneering work, the Neumann bands in iron have given rise to a long-continued controversy. Howe gave elaborate attention to this subject in a long chapter of his well known treatise¹³ under the caption, "The Neumann Bands or Mechanical Twins in Ferrite," which fairly reveals his position in the matter. He demonstrated that these bands habitually occur parallel to planes of form $\{211\}$ (quite in agreement with Osmond, Mügge, Linck and others) but no means of detecting the exact change of orientation across the provisional twinning plane could be found and hence complete proof of crystallographic twinning could not be deduced.

¹⁰ O. Mügge: *Über neuere Strukturflächen an den Krystallen der edelsten Metalle. Jahrb. für Min.* (1899) **2**, 55.

¹¹ *Ztschr. für Kristall.* (1892) **20**, 209.

¹² O. Mügge: *Struktur und einfache Schiebungen des Eisens. Ztschr. anorg. Chem.* (1922) **121**, 68.

¹³ H. M. Howe: *The Metallography of Steel and Cast Iron*. 1916. McGraw-Hill Book Co., Inc., New York.

Conflicting points of view are emphasized by the rather recent work of O'Neill¹⁴ and Rosenhain and McMinn¹⁵ on this general subject. The first of these authors confirms the earlier location of a provisional twinning plane {211} and presents interesting arguments as to the inherent probability of twinning on the ground of a simple translatory mechanism which would readily yield twinned structures. On the other hand, Rosenhain and McMinn were unable to trace slip bands through Neumann lamellae in conformity with the requirements of twinning as seen by them. The present uncertainty concerning the nature of slip bands in ferrite and their unfavorable form would appear to affect seriously an analysis of this sort.

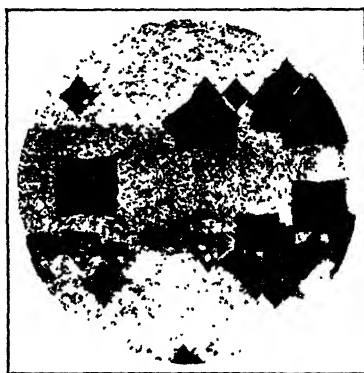


FIG. 2.—ETCHING FIGURES WITHIN AND ADJACENT TO A NEUMANN BAND IN SILICON-FERRITE. (HARNECKER AND RASSOW.)

A striking illustration of the condition which would under favorable circumstances be revealed by slip bands, namely, change of orientation in passing from crystal to twin band, is furnished by the etching figures around and within the Neumann band shown in Fig. 2 from an interesting study of Neumann bands in iron containing 0.6 per cent. nickel and 1.3 per cent. phosphorus by Harnecker and Rassow.¹⁶

The weight of evidence seems to favor the conception that Neumann bands are twin lamellae but the attendant controversy has clearly reached a stage in which no amount of argument founded upon any set of circumstances less vital than the rigorous determination of the crystal structure within and adjacent to the bands can prevail.

¹⁴ H. O'Neill: Deformation Lines in Large and Small Crystals of Ferrite. *Jnl. Iron and Steel Inst.* (1926) **113**, 417.

¹⁵ W. Rosenhain and J. McMinn: The Plastic Deformation of Iron and the Formation of Neumann Lines. *Proc. Roy. Soc.* (1925) **108**, 231.

¹⁶ K. Harnecker and E. Rassow: Aetzfiguren und Zwillingsbildungen in Eisen. *Ztschr. für Metall.* (1924) **16**, 312.

Aluminum

Aluminum seems exceptionally shy in disclosing evidence of twinning. According to C. F. Elam,¹⁷ banded structures are rarely encountered and in two cases investigated by means of the X-rays the plane of junction was not related to the crystal structure of either hypothetical twin component. The crystals did appear to have a dodecahedral plane in common, but as she expresses it "the one could only be derived from the other by turning it upside down and rotating through an angle of 60°."

The absence of the common banded structure which customarily denotes twinning in ordinary metallographic preparations does not prove the absence of twinning either in worked or annealed structures. In the former case twinning in very thin lamellae may accompany or even replace slip and pass unrecognized,¹⁸ while in the latter case, instability at twin boundaries may cause obliteration of the banded structure on annealing.

Zinc and Tin

Zinc and tin are strangely alike in the general circumstances of their twinning although unlike in crystal structure. Both metals twin readily when deformed unless the crystals are properly oriented to deform by pure slip, which according to Mark, Polanyi and Schmid takes place primarily along planes of form $\{0001\}$ and secondarily $\{10\bar{1}0\}$ in the case of zinc,¹⁹ and $\{110\}$ or $\{100\}$ in the case of tin.²⁰ The twin bands, whose formation gives rise to the familiar crackling noise known as the tin or zinc cry, have been described in considerable detail by Mathewson and Phillips²¹ in the case of zinc and Edwards²² in the case of tin. Figs. 3 and 4 show the striking similarity of these structures.

The twinning planes in zinc are of the form $\{10\bar{1}2\}$ according to the crystallographic and X-ray measurements on large crystals reported by Mathewson and Phillips and the twin orientation may be duplicated by rotation through 180° around a twinning axis normal to the twinning plane.

¹⁷ C. F. Elam: *Op. cit.*

¹⁸ Cf. in this connection Rosenhain's reference to Benedicks' views in a discussion of Edwards' paper on Metallic Crystal Twinning. *Jnl. Inst. of Metals* (1915) **14**, 131, and Mügge's observations on translation and twinning in many crystalline substances [*Ztschr. anorg. Chem.* (1922) **121**, 68].

¹⁹ H. Mark, M. Polanyi and E. Schmid: Vorgänge bei der Dehnung von Zinkkristallen I, II and III. *Ztschr. Physik.* (1922) **12**, 58.

²⁰ H. Mark, M. Polanyi: Die Gitterstruktur, Gleitrichtungen und Gleitebenen des Weissen Zinns. *Ztschr. Physik.* (1923) **18**, 75.

²¹ C. H. Mathewson and A. J. Phillips: Plastic Deformation of Coarse-grained Zinc. *Proc. Inst. Metals Div., A. I. M. E.* (1927) 143.

²² C. A. Edwards: Metallic Crystal Twinning by Direct Mechanical Strain. *Jnl. Inst. of Metals.* (1915) **14**, 116.

The early evidence that tin twins along planes of form $\{111\}$ and $\{331\}$ has been discussed by Mügge²³ who determined the angles between

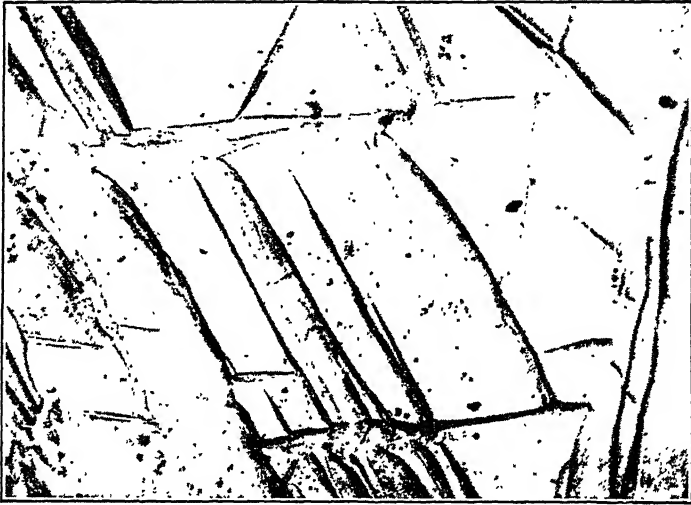


FIG. 3.—TWINS IN ZINC.



FIG. 4.—TWINS IN TIN. (EDWARDS.)

twinning planes in hammered tin crystals and found good agreement with the theoretical requirements for twinning along planes of the latter form in a tetragonal crystal based on the axial ratio 0.3857.

²³ O. Mügge: Über die einfachen Schiebungen am Zinn und seine Zustandsänderung bei 161°. *Zentralbl. für Min.* (1917) 233.

Lead and Nickel

There is ample evidence of abundant twinning in the recorded photomicrographs of the metals lead and nickel in the recrystallized condition. In the case of lead, Humphrey²⁴ has offered evidence of a semiquantitative character, derived from a study of etching figures, that the straight twin boundaries are parallel to octahedral planes.

Fig. 5, showing many straight boundaries and well defined bands indicative of twinning, is of particular interest as it represents the

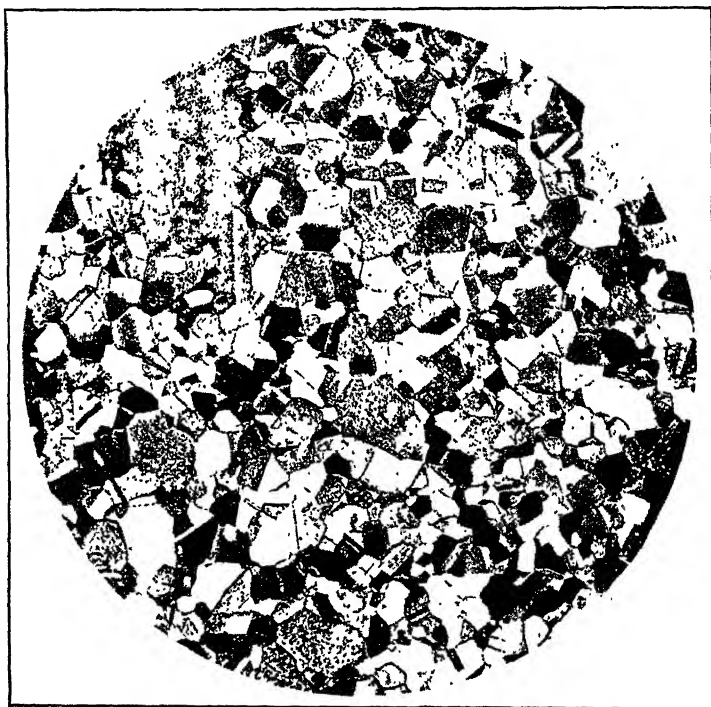


FIG. 5.—TWINS IN SURFACE LAYER OF LEAD CUT WITH A MICROTOME. (F. F. LUCAS.)

recrystallization which occurs spontaneously in the surface layer of lead cut in a microtome and very superficially etched with a solution of hydrogen peroxide and acetic acid. The technique used has been described by F. F. Lucas²⁵ who very kindly prepared this photomicrograph.

GENERAL OCCURRENCE OF TWINNING

If the conditions encountered with the 10 common metals thus far brought into this discussion are fairly representative of metals as a whole,

²⁴ C. W. Humphrey: Effects of Strain on the Crystal Structure of Lead. *Phil. Trans. Roy. Soc.* (1903) 200A, 225.

²⁵ F. F. Lucas: Application of Microtome Methods to the Preparation of Soft Metals for Microscopic Examination. *Proc. Inst. Metals Div., A. I. M. E.* (1927) 481.

we may express in general terms what Mügge²⁶ has written in regard to iron, namely that twinning (customarily) accompanies the reshaping of a metal (by cold working or cold working and annealing) and that without twinning this change of form would indeed occur in a very different manner.

The Face-centered Cubic Metals

In seeking to classify the phenomena of twinning in the entire list of metals, which for convenience may be grouped according to the various types of crystal structure represented,²⁷ I am struck by the lack of crystallographic data in this field. Thus, I have found no complete description of twinning in any of the six metals, calcium, cerium, cobalt, palladium, rhodium and thorium, which together with aluminum, copper, gold, iridium, gamma iron, lead, nickel, platinum and silver, constitute the face-centered cubic group.

A summary of the incomplete information now available is given in Table 1.

TABLE 1.—*Face-centered Cubic Metals*

Metal	Twinning	Composition Plane	Twinning Axis
Aluminum.....	Questionable		
Calcium.....			
Cerium.....	Probable		
Cobalt.....	Probable		
Copper.....	Clearly defined	{ 111 }	[111]
Gold.....	Clearly defined	{ 111 }	[111]
Iridium.....	Clearly defined	{ 111 }	[111]
Gamma iron.....	Clearly defined	{ 111 }	[111]
Lead.....	Clearly defined	{ 111 }	[111]
Nickel.....	Good microscopic evidence		
Palladium.....			
Platinum.....	Clearly defined	{ 111 }	[111]
Rhodium.....			
Silver.....	Clearly defined	{ 111 }	[111]
Thorium.....			

In the case of cerium, the banded structure found by Schumacher and Lucas²⁸ in samples of pure metal may be accepted as qualitative evidence of twinning. This is illustrated in Fig. 6.

²⁶ *Ztschr. anorg. Chem.* (1922) **121**, 68.

²⁷ *International Critical Tables* (1926) **1**, 338.

²⁸ E. E. Schumacher and F. F. Lucas: Photomicrographic Evidence of the Crystal Structure of Pure Cerium. *Jnl. Amer. Chem. Soc.* (1924) **46**, 1167.



FIG. 6.—BANDED STRUCTURE IN PURE CERIUM. (F. F. LUCAS AND E. E. SCHUMACHER.)

Vogel²⁹ has drawn attention to striae believed to be deformational twins found in the surface layers of cobalt, nickel and various alloys after grinding and polishing. So-called annealing twins are familiar elements in the microstructure of nickel and its alloys such as invar, nichrome, permalloy and monel metal, but I do not find that any precise crystallographic study of these twin bands has ever been undertaken.

The Body-centered Cubic Metals

No precise record of twinning has been found in the case of any of the body-centered metals, which, together with alpha iron, are listed in Table 2.

TABLE 2.—*Body-centered Cubic Metals*

Metal	Twinning	Twinning Plane	Twinning Axis
Chromium.	Not observed		
Alpha iron.	Clearly defined	{112}	[111]
Lithium.			
Molybdenum.	Not observed		
Potassium.			
Sodium.			
Tantalum.			
Tungsten.	Not observed		
Vanadium.	Recorded by Groth ^a	{443}?	[443]?

^a P. H. Groth: *Chem. Kristall.* The original paper by Setterberg is inaccessible.

It is rather generally believed that body-centered cubic metals do not form twin crystals. To maintain this point of view it should be necessary to prove that Neumanns in alpha iron are not twin lamellae and that the twins so fully described by McKeehan are merely pseudomorphs of twins originally formed in austenite at a higher temperature. The latter situation would imply that when gamma iron changes to alpha iron certain twin boundary planes of form {111} in the face-centered lattice become converted perhaps without any change of position into composition planes of form {211} between the new crystal and new twin in the body-centered lattice. In order to account fully for the boundary conditions observed in these alpha twins it would be necessary to assume that some other part of an original gamma twin boundary moves into the position of a second boundary plane of form {211} between the same alpha twin and the rest of the crystal.

G. Edmunds at the Hammond Laboratory has been unable to find twin lamellae in the fragments produced by hammering samples of chromium, molybdenum, tantalum, tungsten and vanadium.

²⁹ R. Vogel: Über Zwillingsbildung in den Oberflächen Schichten von Metallen infolge Kaltbearbeitung. *Ztschr. anorg. Chem.* (1921) **117**, 271.

Nothing is known concerning the behavior of the alkali metals in respect to twinning.

The Hexagonal Metals

The metals allied to zinc, except mercury, in the hexagonal close-packed group have been studied by Mathewson and Phillips,³⁰ who find twinning on planes of form $\{10\bar{1}2\}$ in every case quite in conformity with the twinning behavior of zinc (see Table 3). No other metal in this group has been studied from the present point of view.

TABLE 3.—*Hexagonal Close-packed Metals*

Metal	Twinning	Twinning Plane	Twinning Axis	Axial Ratio
Beryllium.....	Clearly defined	$\{10\bar{1}2\}$	$[10\bar{1}2]$	1.58
Cadmium.....	Clearly defined	$\{10\bar{1}2\}$	$[10\bar{1}2]$	1.89
Cerium.....				1.62
Cobalt.....				1.633
Hafnium.....				1.64
Magnesium.....	Clearly defined	$\{10\bar{1}2\}$	$[10\bar{1}2]$	1.624
Osmium.....				1.59
Ruthenium.....				1.59
Titanium.....				1.59
Zinc.....	Clearly defined	$\{10\bar{1}2\}$	$[10\bar{1}2]$	1.86
Zirconium.....				1.59

Other Metals of Known Crystal Structure

The remaining metals of known crystal structure are grouped in Table 4. Twinning in antimony and bismuth by pressure was described

TABLE 4.—*Miscellaneous Metals*

Metal	Twinning	Twinning Plane	Twinning Axis	Form of Lattice
Antimony.....	Clearly defined	$\{01\bar{1}2\}$ ^a	$[01\bar{1}2]$	Hex.-rhombohedral
Bismuth.....	Clearly defined	$\{01\bar{1}2\}$ ^a	$[01\bar{1}2]$	Hex.-rhombohedral
Germanium.....				Diamond cubic
Indium.....	Microscopic evidence	^b		Face-centered tetragonal
Tellurium.....				Hex.-rhombohedral
Tin (gray).....				Diamond cubic
Tin (white).....	Clearly defined	$\{331\}$ or $\{111\}$?	$[331]$ or $[111]$?	Tetragonal of special form

^a See footnote 31.

^b See footnote 32.

³⁰ C. H. Mathewson and A. J. Phillips: Twinning in Beryllium, Magnesium, Zinc and Cadmium. *Proc. Inst. Metals Div., A. I. M. E.* (1928) 445.

by Mügge more than 40 years ago.³¹ Carpenter and Tamura³² have photographed twin bands of the usual appearance in a deformed and annealed sample of indium. Germanium and tellurium remain untouched in this field of investigation.

Metals of Unknown Crystal Structure

I have found no record of twinning either qualitative or quantitative in any of the remaining metals: actinium, columbium, gallium, manganese, mercury, niobium, polonium, radium, scandium, thallium, uranium, yttrium and rare earth metals.

Summary

It may be said in concluding this survey that twinning in one form or another is an ordinary feature of the microstructure of all the familiar metals except aluminum. It is known to occur in some of the other metals but comparatively few have been investigated. Obviously the absence of twinning cannot be postulated in any of the negative cases. For example, it has not been demonstrated, although it may appear improbable, that twinning has no part in the fragmentation and fibering of tungsten by swaging, as described by Smithells.³³

Twinning therefore is prevalent in metals and consequently a factor of great influence on the shape, size, and particularly the orientation of the crystalline particles formed when metals are forced to change their shape by mechanical treatment and brought to various temper conditions by annealing.

SOME OBSERVATIONS ON TWINNING AND RECRYSTALLIZATION IN WROUGHT METALS OF THE HEXAGONAL AND CUBIC GROUPS

This includes most of the metals. The experimental work is restricted to one metal in each group. Only in the case of zinc are the observations systematic from a crystallographic point of view and even in this case the work is incomplete in many directions. Iron is discussed quite briefly on the basis of Krivobok's experiments.³⁴ The observations on copper are for the most part fragmentary and circumstantial. Nevertheless, there is a continuity of circumstance and an undercurrent of analogy pointing to a general group of postulates or provisional assertions, which may at least stir up an amount of useful discussion.

³¹ O. Mügge. Über künstliche Zwillingsbildung durch Druck am Antimon, Wis-
muth und Diopsid. *Neue Jahrb. für Min.* (1886) **1**, 183.

³² H. C. H. Carpenter and S. Tamura: The Formation of Twinned Metallic Crys-
tals. *Proc. Roy. Soc.* (1926-27) **113A**, 161.

³³ C. J. Smithells: Tungsten. 1926. Chapman & Hall, London.

³⁴ V. N. Krivobok: A Photomicrographic Study of the Process of Recrystallization
in Certain Cold-worked Metals. A. I. M. E. *Pamphlet* No. 1557-E (1926).

Rolled Zinc

With adaptable single crystals at hand, it appears to be a simple matter to plan and execute a set of experiments in which various selected orientations are deformed plastically by rolling, and critically examined under the microscope from time to time as they change structurally during the process.

In practice, difficulties were found which have thus far limited the work to the rolling of thin sections bounded top and bottom by basal cleavages. Chilled flat-metal single crystals about $\frac{1}{4}$ in. thick with their basal planes nearly perpendicular to the surface could be broken quite easily along these planes by simply bending or forcing a sharp blade into the metal in the desired region. In this way sections down to 1 mm. thick were produced without twinning except in a few localized areas.

In any small-scale laboratory study of a rolling process the advantage of having a specimen with two perfectly flat and parallel surfaces cannot be overestimated. It is, of course, impossible to retain this advantage

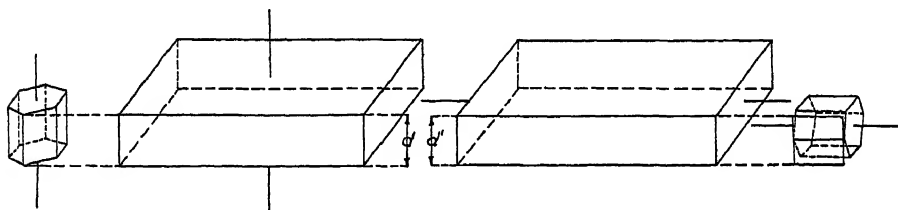


FIG. 7.—APPROXIMATE CHANGE IN ORIENTATION DUE TO SIMPLE TWINNING IN ZINC.

in specimens of other orientations, which must be cut to specifications from the original single crystal. The operations of cutting, grinding and polishing such small samples, even when conducted with extreme care, produce an amount of twinning which materially changes the reaction of the material to rolling or other designated forms of stress.

A great many experiments were made with cleavage sections of different thickness rolled various amounts either continuously or in stages.

Only the most general conclusions will be presented here, together with such illustrations as seem necessary to give them force and clarity.

If we imagine the atoms in a basal cleavage section of thickness d to be completely rearranged by twinning along one set of planes of form $\{10\bar{1}2\}$ without changing the horizontal plane of the section, the vertical hexagonal axis assumes a position only 4° removed from the horizontal and the thickness decreases 7.21 per cent.

This is nearly equivalent to turning the unit hexagonal prism on its side as shown in Fig. 7 so that the thickness which was originally proportional to the height ($d = 1.862$) becomes nearly proportional to the distance between prismatic faces ($d' = \sqrt{3}$).

If the twinning occurs in different parts of the specimen along different planes of form $\{10\bar{1}2\}$ —for example, $(01\bar{1}2)$, $(0\bar{1}12)$, $(1\bar{1}02)$, and $(\bar{1}102)$ —the hexagonal axis will assume different directions, 60° apart, in planes 4° from the horizontal and the reduction in thickness may be somewhat less than in the simple case owing to redistribution of the strain which is theoretically in evidence at twin boundaries when we follow a simple geometrical plan of bringing the atoms up to a common composition plane.

With large portions of the specimen dominated by a single set of twinning planes this effect would of course be negligible, but with an involved interlacing of minute twins it might be considerable.

The conditions observed by K. R. Van Horn in the Hammond Laboratory experiments favor the simple interpretation. Many basal cleavage sections rolled to a reduction of 6 to 7 per cent. contained

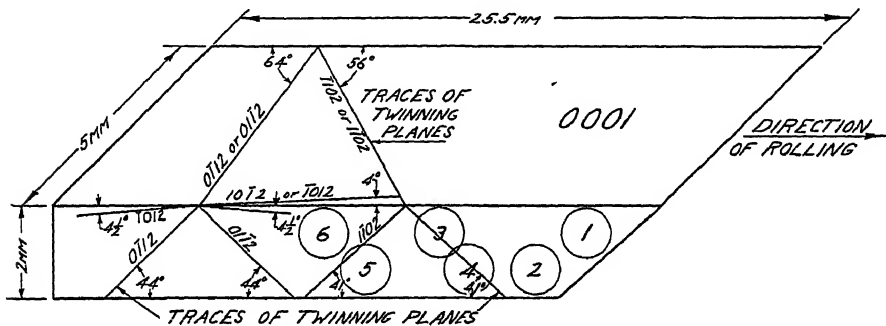
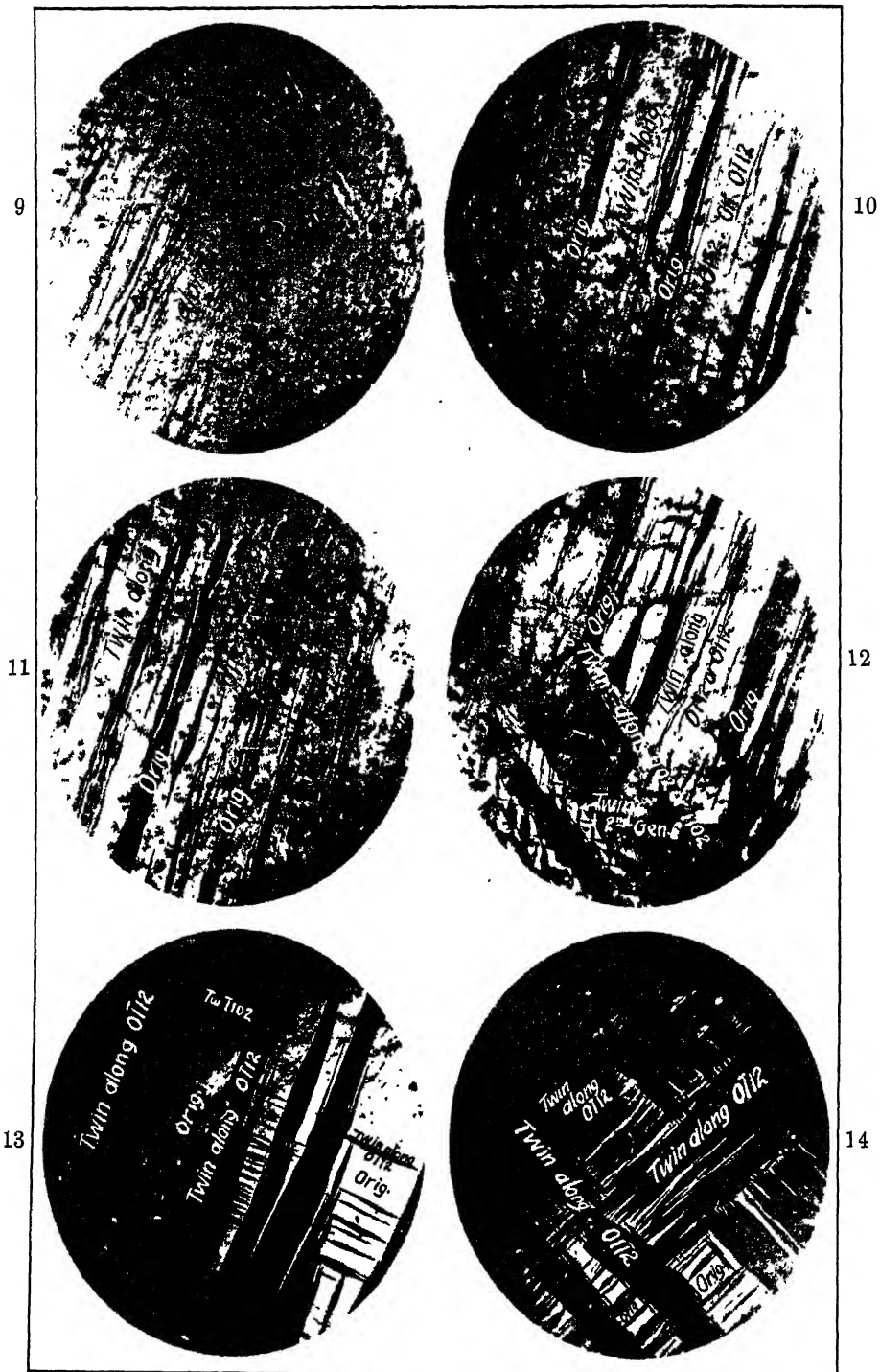


FIG. 8.—POSSIBLE TWINNING PLANES AND SPOTS PHOTOGRAPHED IN A ROLLED BASAL CLEAVAGE SECTION OF A ZINC CRYSTAL.

large regions wholly converted to a single preferred twin orientation. The prevailing conditions are represented by the following analysis of one specimen. The dimensions and orientation of this specimen are shown in Fig. 8.

The thickness was reduced 6.98 per cent. by two passes through a small set of hand-operated rolls. The specimen was then dipped in moderately concentrated nitric acid and examined on both rolled surfaces under the microscope. In this work plane, polarized light was passed through a vertical illuminator and, after reflection from the surface of the specimen, through an analyzing nicol prism which was usually set to extinguish light vibrating in the original (nonrotated) plane of polarization. On rotating the stage, contrast could be developed between the various structure elements and twin bands could be identified either by measuring the angles between various twinning planes or determining the positions of complete parallel extinction. For quantitative work the former method was preferred as it was less influenced by surface inequalities due to variable etching, wavy rolling, etc. Unaltered basal



surfaces which normally should be optically isotropic actually remained uniformly dark during complete rotation through 360° .

Figs. 9, 10, 11 and 12 represent several portions of the top and bottom surfaces photographed between crossed nicols and marked for identification of the structure elements shown. In all of these photos only small remnants of the original base $\{0001\}$ are visible. *Almost the entire substance of the crystal has been transformed by twinning into another crystal.* Bands or patches formed by twinning along all possible planes of form $\{10\bar{1}2\}$, six in number, are combined in the new crystal. The distribution varies in different parts of the crystal but it can be stated that $(0\bar{1}12)$ and $(01\bar{1}2)$ are strongly preferred planes of twinning in this specimen while $(10\bar{1}2)$ and $(\bar{1}012)$ merely contributed a few bands along the edge due to bending stresses set up when the original cleavages were made.³⁵

Figs. 9 and 10 show that either $(0\bar{1}12)$ or $(01\bar{1}2)$ are preferred planes at the top of the specimen and Figs. 11 and 12 that a similar condition exists at the bottom surface. These planes come nearest into line with the axis of revolution of the rolls.

The nature of this preference is revealed by examining the sides, which had to be smoothed and polished carefully with fine abrasive to remove the roughness due to rolling and at the same time retain the original structure. The beveled end, which was a natural feature of the specimen, entered the rolls first, as shown in Fig. 8. At the spot marked 1, about halfway down the beveled edge, less than half of the metal (shown white in Fig. 13) retains its original orientation and the rest (shown dark in Fig. 13) is twinned in general along the $(0\bar{1}12)$ planes. A few narrow bands twinned along $(01\bar{1}2)$ planes may be seen at the right and in the upper center a band twinned along $(\bar{1}102)$.

At the spot marked 2, just beyond the beveled edge, represented by Fig. 14, very little of the original crystal remains and the metal is prin-

³⁵ The identity of these bands is fixed by tracing them across the edge into the upper or lower surface. It may be noted that the pair $(0\bar{1}12)$ and $(01\bar{1}2)$ can easily be distinguished from the pair $(\bar{1}102)$ and $(1\bar{1}02)$ on the surface and the pair $(0\bar{1}12)$ and $(1\bar{1}02)$ from the pair $(01\bar{1}2)$ and $(\bar{1}102)$ on the edge.

FIG. 9.—ZINC IN POLARIZED LIGHT. CROSSED NICOLS. $\times 100$. TOP SURFACE IN FIG. 8.

FIG. 10.—ZINC IN POLARIZED LIGHT. CROSSED NICOLS. $\times 100$. TOP SURFACE IN FIG. 8.

FIG. 11.—ZINC IN POLARIZED LIGHT. CROSSED NICOLS. $\times 100$. BOTTOM SURFACE IN FIG. 8.

FIG. 12.—ZINC IN POLARIZED LIGHT. CROSSED NICOLS. $\times 100$. BOTTOM SURFACE IN FIG. 8.

FIG. 13.—ZINC IN POLARIZED LIGHT. CROSSED NICOLS. $\times 100$. STRUCTURE AT (1) IN FIG. 8.

FIG. 14.—ZINC IN POLARIZED LIGHT. CROSSED NICOLS. $\times 100$. STRUCTURE AT (2) IN FIG. 8.

(Reduced to $\frac{3}{4}$ original scale, original magnification given.)

cipally composed of twins along $(0\bar{1}12)$ coming down from the top and $(01\bar{1}2)$ coming up from the bottom.

This condition is typical of the structure at other locations with the added qualification that the two varieties of preferred twins occur in variable quantities with the first always in excess. Thus Fig. 15, repre-

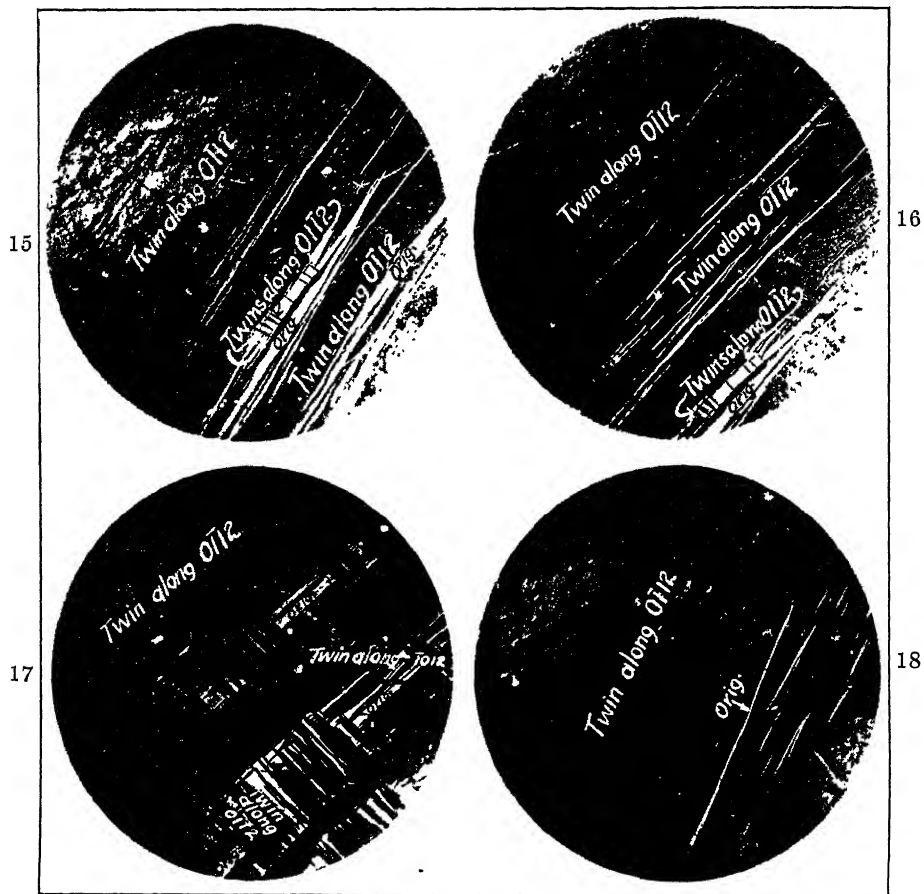


FIG. 15.—ZINC IN POLARIZED LIGHT. CROSSED NICOLS. $\times 100$. STRUCTURE AT (3) IN FIG. 8.

FIG. 16.—ZINC IN POLARIZED LIGHT. CROSSED NICOLS. $\times 100$. STRUCTURE AT (4) IN FIG. 8.

FIG. 17.—ZINC IN POLARIZED LIGHT. CROSSED NICOLS. $\times 100$. STRUCTURE AT (5) IN FIG. 8.

FIG. 18.—ZINC IN POLARIZED LIGHT. CROSSED NICOLS. $\times 100$. STRUCTURE AT (6) IN FIG. 8.

(Reduced to $\frac{3}{4}$ original scale, original magnification given.)

sending the spot marked 3 near the top of the specimen, shows only traces of the second preferred twin; Fig. 16, a spot (4) near the bottom, about the same; Fig. 17, another spot (5) near the bottom, considerably more of the second twin and finally Fig. 18, a spot near the upper center,

virtually a complete transformation by twinning along $(0\bar{1}12)$ planes. The last figure also represents the structure of the rest of the material visible on this section.

The opposite side of the specimen shows a more general participation of the other two twinning planes $(1\bar{1}02)$ and $(\bar{1}102)$. Various conditions of preferred twinning along the four predominant planes are observed on examination of different cleavage specimens from the same crystal. When the direction of rolling is exactly parallel to the edge, the planes $(0\bar{1}12)$ and $(01\bar{1}2)$ are inclined 26° and the planes $(1\bar{1}02)$ and $(\bar{1}102)$ 34° to the axis of revolution of the rolls. Thus an inconsiderable displacement of 4° in the right direction as the specimen passes

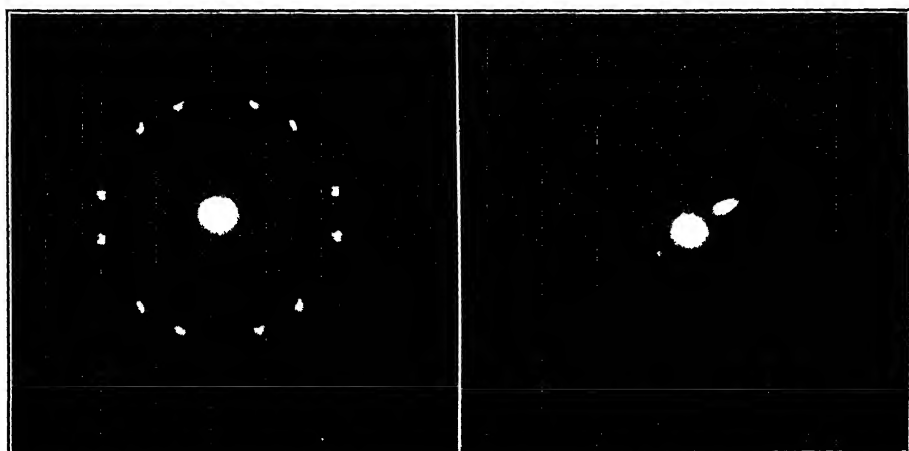


FIG. 19.—LAUE PATTERN FROM UNTWINNED BASAL CLEAVAGE.

FIG. 20.—LAUE PATTERN FROM BASAL CLEAVAGE REDUCED 6 TO 7 PER CENT. IN THICKNESS BY COLD ROLLING.

through the rolls should bring these two sets of planes into equivalent action.

This condition was actually discovered by R. M. Bozorth of the Bell Telephone Laboratories in a Laue pattern³⁶ of one of these specimens which had been reduced 6.6 per cent. in thickness by one pass through the rolls. Another specimen rolled almost exactly the same amount furnished a pattern showing one predominant twin. The pattern from an untwinned cleavage is shown in Fig. 19 and one of the altered patterns in Fig. 20. So much of the radiation was absorbed by these specimens, which were about $\frac{1}{2}$ mm. thick, that only the most strongly reflecting planes are recorded in the halftones. These are the planes of form $\{12\bar{3}1\}$ in the original crystal responsible for twelve spots equidistant from the center and basal planes from four twins inclined 86° to the

³⁶ General radiation from a tungsten target was employed in this and similar experiments and the beam was directed at right angles to the cleavage surface.

cleavage surface represented by one bright spot and three much fainter spots of varying intensity near the center.

It is observed that the spots from the original crystal are scarcely discernible in Fig. 20 while the dominant intensity of the spot just above and to the right of the spot formed by the undeviated beam indicates that the crystal is principally composed of material twinned along the preferred plane ($0\bar{1}12$). The original negative also revealed faint spots which could be referred to planes of form $(21\bar{3}0)$ and $(21\bar{3}1)$ in the twin. The elongated and generally ragged appearance of all diffraction spots from rolled samples is attributed to distortion associated



FIG. 21.

FIG. 22.

FIG. 21.—ZINC IN POLARIZED LIGHT. CROSSED NICOLS. $\times 100$. BASAL CLEAVAGE SECTION REDUCED 18 TO 19 PER CENT. IN THICKNESS BY COLD ROLLING. REACTION BETWEEN TWIN BANDS (0112) AND (1102) THE LATTER ROTATED TO A POSITION OF EXTINCTION. NARROW INTERNAL TWIN BANDS OF SECOND GENERATION VISIBLE.

FIG. 22.—ZINC IN POLARIZED LIGHT. CROSSED NICOLS. $\times 100$. BASAL CLEAVAGE SECTION REDUCED 18 TO 19 PER CENT. IN THICKNESS BY COLD ROLLING. REACTION BETWEEN TWIN BANDS (0112) AND (1102) THE FORMER ROTATED TO A POSITION OF EXTINCTION. NARROW TWIN BANDS OF SECOND GENERATION VISIBLE.

(Reduced to $\frac{3}{4}$ original scale, original magnification given.)

with the twinning and the accompanying basal slip revealed by slip bands in some of the twinned portions along an edge which had been carefully polished and lightly etched before rolling.

When these cleavage sections are rolled beyond the amount required for complete twinning, further extension must take place by some slipping process. Planes of the two forms representing the closest atomic packing, viz., $\{0001\}$ nearly vertical and $\{10\bar{1}0\}$ nearly horizontal, seem to be unfavorably located for slip, but the polished and etched side of the specimen described in Fig. 8 showed numerous faint striae parallel to both of these planes when it was rolled to a further slight reduction. In addition some of the twin bands along the principal

preferred plane ($0\bar{1}12$) contained striae inclined about 67° to the edge, not greatly different from the value 64° which would represent the traces of twinning planes yielding certain twins of a second generation actually observed in the primary bands (d in Fig. 25). In some places curved striae were observed. It is concluded that there is a complication of slip and twinning along several sets of planes accompanied by a certain amount of distortion.

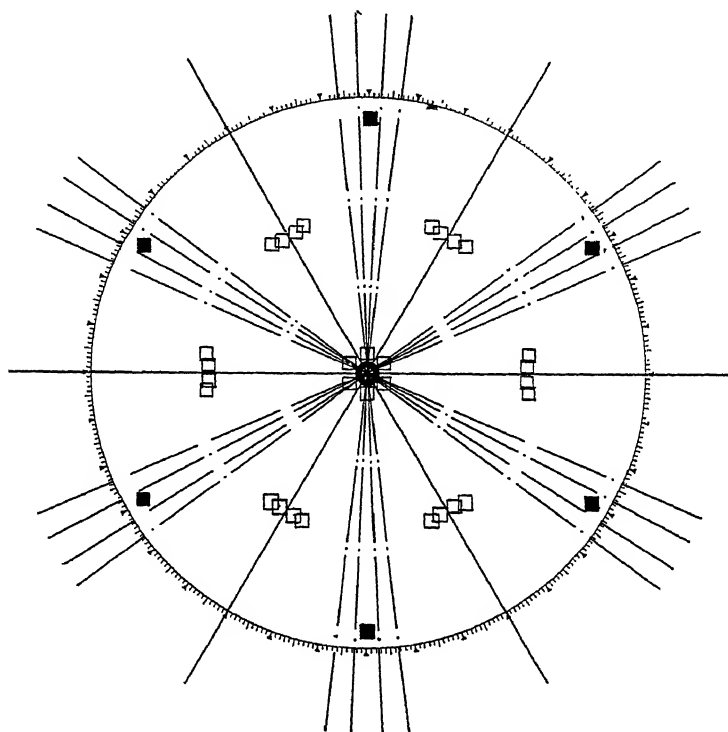


FIG. 23.—STEREOGRAPHIC PROJECTION OF BASAL PLANES IN A ZINC CRYSTAL AND ALL POSSIBLE TWINS THROUGH TWO GENERATIONS, SHOWING TRACES OF THESE PLANES IN THE PLANE OF THE PROJECTION.

- = pole of basal plane in the original crystal.
- = poles of basal planes in twins of the first generation.
- = poles of basal planes in twins of the second generation.

The surface structure of a specimen reduced 18.9 per cent. in thickness in three passes through the rolls furnishes considerable evidence concerning the nature of recrystallization. The customary angular intergrowths of twin bands sometimes including lamellae of the original crystal develop by interchange of material across the boundaries into more or less rounded dendritic forms, with the frequent occurrence of detached lobes or branches. This effect, which is illustrated in Figs. 21 and 22, taken between crossed nicols, represents recrystallization in the limited sense that there has been a granulation and redistribution but not a

pronounced reorientation of the material. It is commonly regarded as spontaneous recrystallization during working in conformity with the well-known fact that cold-rolling of zinc substantially at room temperature produces the type of alteration which is usually associated with the hot-working of metals. In taking Fig. 21, the stage was rotated to bring the central dendritic patch with its appendages into a position of complete extinction. A simple analysis with the aid of a stereographic projection of the basal planes in the crystal and all possible twins through two generations, shown in Fig. 23, identifies this patch as an original (preferred) twin along $(1\bar{1}02)$ planes; *viz.*, a line perpendicular to a line joining the pole of a basal plane, as reorientated by twinning, with the



FIG. 24.—ZINC IN POLARIZED LIGHT. CROSSED NICOLS. $\times 100$. BASAL CLEAVAGE SECTION REDUCED 18 TO 19 PER CENT. IN THICKNESS BY COLD ROLLING. TWIN BANDS OF THE SECOND GENERATION INTERSECTING AT 88° IN AN ORIGINAL TWIN ALONG $(0\bar{1}12)$ PLANES VIEWED IN THE CLEAVAGE SURFACE.

(Reduced to $\frac{3}{4}$ original scale, original magnification given.)

center of the projection gives one of the four positions of extinction when this twin is rotated in the plane of the original basal cleavage. Fig. 22 represents the same location on the specimen rotated to produce extinction of the conspicuous intergrowth. This proves to be a twin parallel to $(0\bar{1}12)$ planes.

Complex twinning was expected in the hard-rolled specimens and in the present case is particularly well represented by two varieties of twins belonging to a second generation derived from original twin bands along $(0\bar{1}12)$ planes. These intersect approximately at right angles as shown in Fig. 24 and the poles of their corresponding twinning planes are located at *a* and *b* in Fig. 25, which is a projection in the original cleavage plane of the poles of all possible twinning planes through two generations. The lines *a'* and *b'* represent the traces of these planes in the original base and the angle between them is 88° . The poles representing other twins

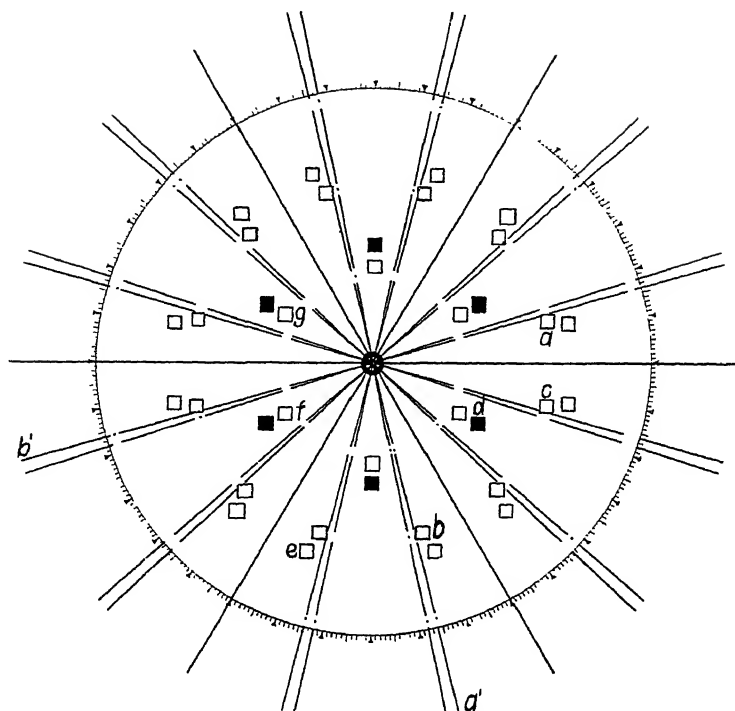


FIG. 25.—STEREOGRAPHIC PROJECTION OF TWINNING PLANES IN A ZINC CRYSTAL, AND ALL POSSIBLE TWINS THROUGH TWO GENERATIONS, SHOWING TRACES OF THESE PLANES IN THE ORIGINAL BASAL PLANE, WHICH IS THE PLANE OF THE PROJECTION.

- = pole of basal plane in the original crystal.
- = poles of twinning planes yielding twins of the first generation.
- = poles of twinning planes yielding twins of the second generation.

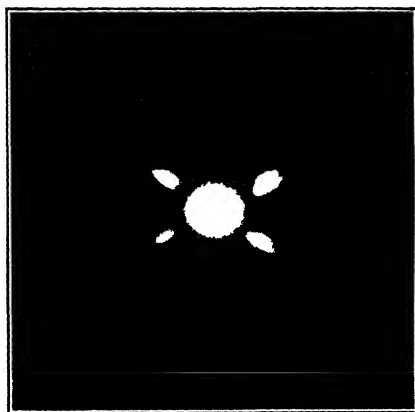


FIG. 26.—LAUE PATTERN OF BASAL CLEAVAGE SECTION REDUCED 18 TO 19 PER CENT. IN THICKNESS BY COLD ROLLING.

belonging to a second generation observed in this specimen are marked *c*, *d*, *e*, *f* and *g* in Fig. 25.

The Laue pattern obtained from this specimen is given in Fig. 26. An outer grouping of faint spots (not visible in the reproduction) corresponding to the prominent spots in Fig. 19 proves that some of the

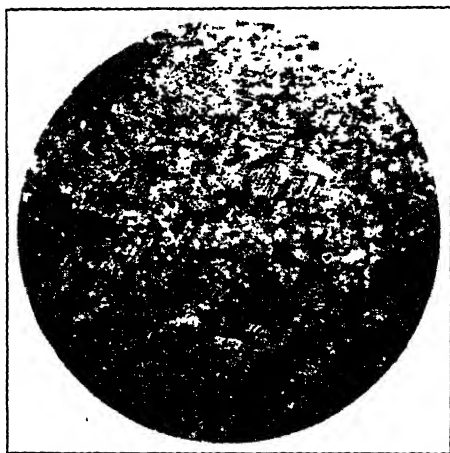


FIG. 27.—ZINC IN POLARIZED LIGHT. BASAL CLEAVAGE SECTION REDUCED 50 PER CENT. IN THICKNESS BY COLD ROLLING. RECRYSTALLIZATION WITH DIVERSITY OF ORIENTATION.

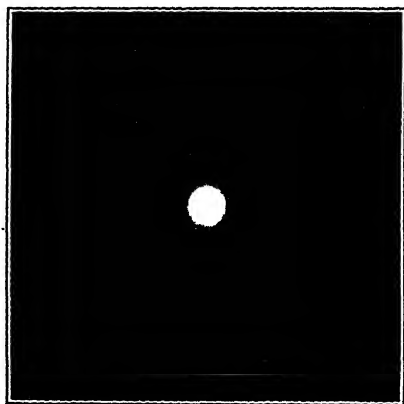


FIG. 28.—LAUE PATTERN OF BASAL CLEAVAGE SECTION REDUCED 50 PER CENT. IN THICKNESS BY COLD ROLLING.

original crystal still remains. This may indicate that a concentration of impurities in certain parts of the crystal has prevented the atomic movements required for twinning. An interesting discussion of effects of this character as related to pure slip has been contributed this year by M. J. Buerger.³⁷ The extreme faintness of spots at the extremities of a

³⁷ M. J. Buerger: The Cause of Translation Striae and Translation Strain-hardening in Crystals. *Proc. Inst. Metals Div., A. I. M. E.* (1928) 375.

vertical diameter through the central group indicates that practically no twins had developed along $(10\bar{1}2)$ and $(\bar{1}012)$ planes in the cubic millimeter of material traversed by the X-ray beam. The prominence of the other four spots signifies that the specimen is for the most part

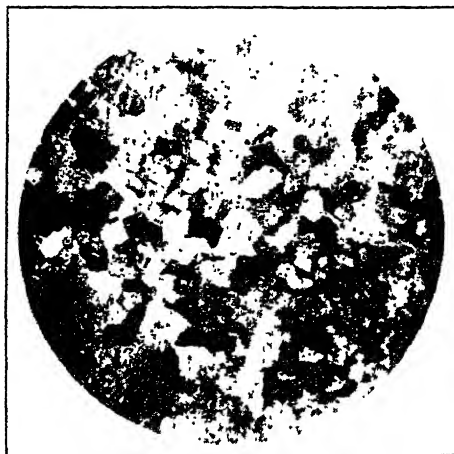


FIG. 29.—ZINC IN POLARIZED LIGHT. BASAL CLEAVAGE SECTION REDUCED 50 PER CENT. BY COLD ROLLING AND ANNEALED 10 MIN. AT 250°C . RECRYSTALLIZATION AND GRAIN GROWTH.

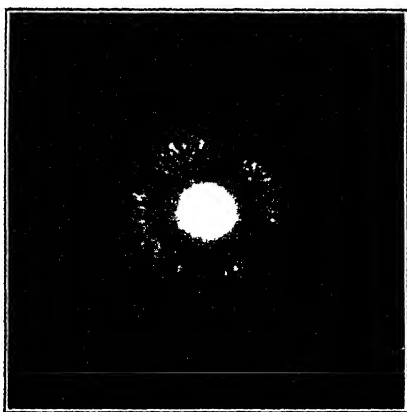


FIG. 30.—LAUE PATTERN OF BASAL CLEAVAGE. SECTION REDUCED 50 PER CENT. IN THICKNESS BY COLD ROLLING AND ANNEALED 10 MIN. AT 250°C .

composed of twins formed along the four planes, $(0\bar{1}12)$, $(01\bar{1}2)$, $(1\bar{1}02)$, and $(\bar{1}102)$.

Part of the etched surface of a specimen reduced 50 per cent. in thickness by cold rolling is shown in Fig. 27. It is probable that the prevailing fine granulation is a result of reaction between descendent twin lamellae just as the coarse granulation in an earlier stage came from the original

twins. This might result in a sufficient diversity of orientation to account for the Laue pattern which is reproduced in Fig. 28. The original crystal seems to have disappeared but the positions of the six original twins are marked by selective groupings of the diffraction spots in the central ring. These details are unsatisfactory in the reproduction.

A piece of this specimen was annealed for ten minutes at 250° C. The microstructure given in Fig. 29 and the Laue pattern in Fig. 30 reveal grain growth but at present it is not possible to decide whether any new orientations have been produced, apart from those which would approximate the requirements of complex twinning. It may be stated that a full complement of twins through only three generations would permit no angle greater than 5° to separate adjacent traces of planes of form {0001} in the original cleavage surface.

Hammered Iron and Rolled Beta Brass

Krivobok's experiments³⁸ demonstrate that the recrystallization effected by annealing pure electrolytic iron or iron containing about 1.75

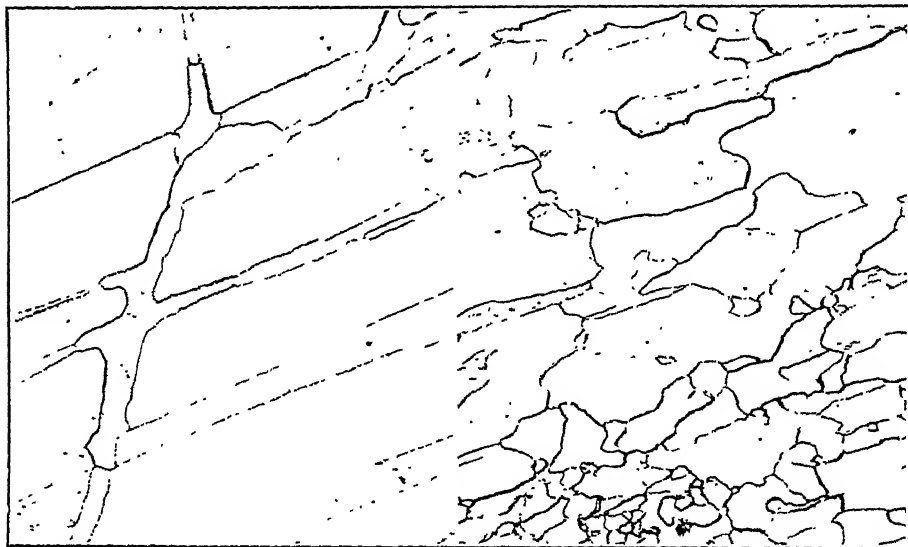


FIG. 31.—AN EARLY STAGE OF RECRYSTALLIZATION IN SILICON-FERRITE PRODUCED BY ANNEALING AFTER COLD-HAMMERING. (KRIVOBOK.)

FIG. 32.—A LATER STAGE OF RECRYSTALLIZATION IN SILICON-FERRITE. (KRIVOBOK.)

per cent. of silicon, previously reduced 25 per cent. in thickness by cold hammering, can be definitely traced to numerous Neumann bands which he regarded as the principal seat of strain in the cold-worked material, without attempting to define their constitution. An assumption that they are mechanical twins with distortion at the boundaries would favor

³⁸ V. N. Krivobok: *Op. cit.*

the deductions drawn from these experiments. Two of the many excellent photographs included in the paper are reproduced for the purpose of illustrating some of the significant transformations observed.

In Fig. 31 a band near the left-hand margin has developed into a dendritic growth by invading the original crystal on both sides and feeding along several intersecting bands. The similarity between this process and the reaction between twin bands in zinc is quite evident. In the present case the original bands are narrow intrusions in a predominant volume of the original crystal and nothing is known concerning the orientations of the various components either before or after the transformation. That the entire substance of the crystal may be irregularly partitioned by growth emanating from these narrow bands is evident on inspection of Fig. 32 in which the directional influence of the bands can be seen in spite of their partial disintegration and subsequent growth in various directions.

There has been a general failure to recognize any participation of twinning in the general process of deformation and recrystallization as observed in iron of the usual fine-grained structure. Chappell,³⁹ who carefully studied recrystallization in iron wire containing 0.07 per cent. carbon, in describing the first stage writes as follows: "These granular markings (representing the first change on annealing), indicating disintegration and incipient recrystallization of the deformed ferrite crystals, very often take the form of roughly parallel and curved bands crossing the crystals obliquely to the longitudinal axis."

How much of this failure may be attributed to the extremely minute size of the strain markings and the first elements of recrystallization, making it difficult to recognize any eventual participation of twins in the process and how much to the alleged circumstance that Neumann bands are produced only by special forms of deformation cannot be predicted. It is perhaps significant that these special conditions, namely, shock in coarse-grained metal, would particularly favor the production of twin bands large enough to be recognized without difficulty. So keen an observer as L. W. McKeehan⁴⁰ has suggested, as an alternative to the columnar slipping proposed by Taylor and Elam⁴¹ to account for curved slip bands in iron, twinning on a very fine scale with zigzag boundaries which would be possible on account of the peculiar relation of twins along planes of form $\{211\}$ in a body-centered cubic structure.

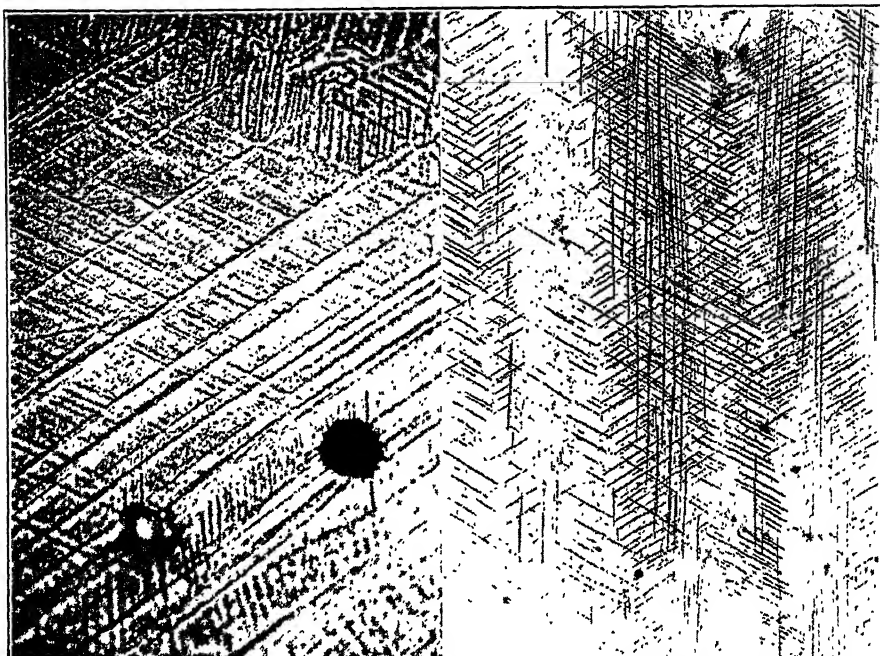
Good photomicrographs illustrating mechanical twinning in rolled beta brass may be found in Miss Clark's paper on heat treatment of 60-

³⁹ C. Chappell: The Recrystallization of Deformed Iron. *Jnl. Iron and Steel Inst.* (1914) **89**, 460.

⁴⁰ L. W. McKeehan: *Op. cit.*

⁴¹ G. I. Taylor and C. F. Elam: The Distortion of Iron Crystals. *Proc. Roy. Soc.* (1926) **112A**, 337.

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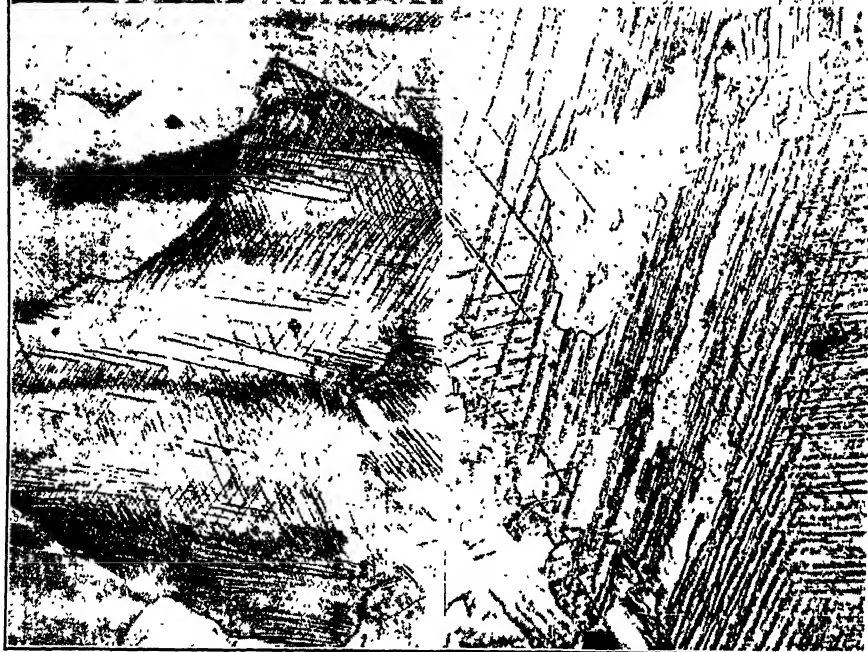


FIG. 33.—DEFORMATIONAL STRIAE IN TIN BRONZE. $\times 1000$.

FIG. 34.—DEFORMATIONAL STRIAE IN A SINGLE CRYSTAL OF ALPHA BRASS. $\times 10$

FIG. 35.—DEFORMATIONAL STRIAE IN ALPHA BRASS. $\times 75$.

FIG. 36.—A VERY EARLY STAGE OF RECRYSTALLIZATION IN ALPHA BRASS. $\times 50$
(Reduced to $\frac{1}{5}$ original scale, original magnification given.)

40 brass.⁴² In a discussion of that paper⁴³ I pointed out the intimate connection between twinning and recrystallization observed in a pure beta brass with about 53 per cent. of copper. There are many points of similarity in the recrystallization of alpha iron and beta brass and it is noteworthy that both possess the body-centered cubic type of structure.⁴⁴

Copper and Its Alpha Solutions

Bands of detectable width at once suggesting twin lamellae are not infrequently observed in microsections of strained copper, brasses and bronzes. As in the other examples of mechanical twinning, deformation by impact is especially productive of these markings, which belong to the group designated Non-effaceable Deformation Figures by Osmond and Cartaud,⁴⁵ and correspond to the ones found by these authors in face-centered cubic iron parallel to planes of form $\{111\}$.

A. J. Phillips, working in the Hammond Laboratory, has recently attempted to trace the relationship between these markings and the structure elements which develop on annealing. Deformation figures of striking appearance were found in the altered surface layers of metal shaped by rough grinding with an emery wheel. A sample of copper containing 12.5 per cent. of tin, previously homogenized, developed the regular pattern shown in Fig. 33. Part of a single crystal in cast brass containing 70 per cent. of copper revealed a quite variable distribution of three intersecting sets of bands according to the pattern shown in Fig. 34. Various combinations of bands occur with variable spacing in different parts of the large crystal shown in Fig. 35. Curvature of the bands indicating distortion is also prominent in certain parts of the specimen.

The best proof that these bands are twin bands is that, on annealing, groups of narrow parallel bands merge to form twin bands of ordinary appearance. A very early stage in this process is shown in Fig. 36, a later stage in Fig. 37 and a more perfected stage in Fig. 38.

Theoretically there should be no strain at a simple twin boundary in the face-centered cubic lattice but this is neither true of the extremities nor of the intersections with other bands. Consequently parallel bands may grow into one another from the ends, as is often observed even in the case of the more gradual changes in somewhat stabilized structures at elevated temperatures. This effect is illustrated particularly well by the central group of bands shown in Fig. 39. The selective infiltration

⁴² F. H. Clark: A Study of the Heat Treatment, Microstructure and Hardness of 60-40 Brass. *Proc. Inst. Metals Div., A. I. M. E.* (1927) 276.

⁴³ *Op. cit.*, 299.

⁴⁴ A recent investigation of the structure of beta brass by Arthur Phillips and L. W. Thelin was reported in the *Journal of the Franklin Institute*, September, 1927.

⁴⁵ F. Osmond and G. Cartaud: *Op. cit.*

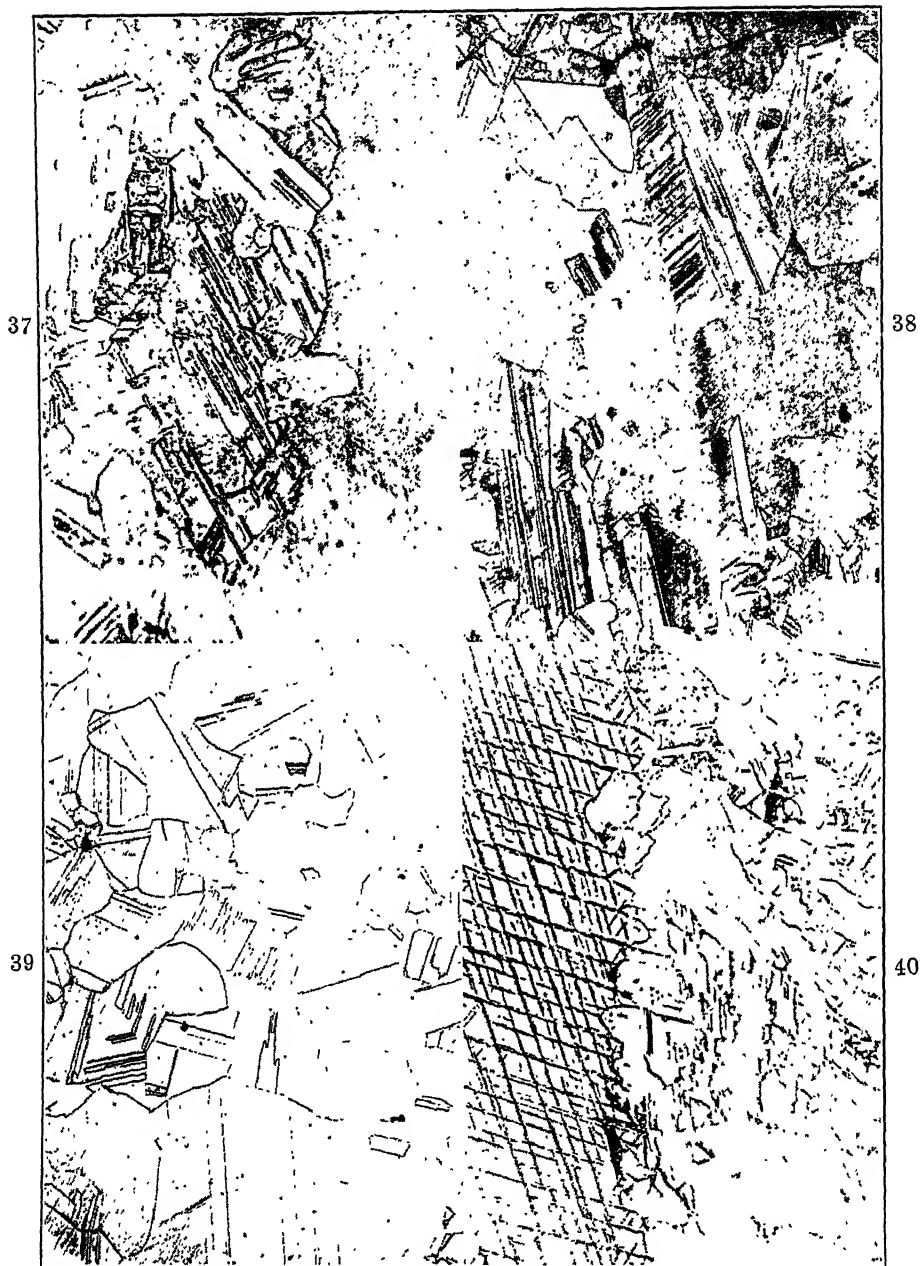


FIG. 37.—AN EARLY STAGE OF RECRYSTALLIZATION IN ALPHA BRASS. $\times 500$.

FIG. 38.—AN ADVANCED STAGE OF RECRYSTALLIZATION IN ALPHA BRASS. $\times 100$.

FIG. 39.—COALESCENCE OF TWIN BANDS BY END GROWTH. $\times 100$.

FIG. 40.—RECRYSTALLIZATION ALONG PATHS PROVIDED BY INTERSECTING TWIN BANDS. $\times 500$.

(Reduced to $\frac{4}{5}$ original scale, original magnification given.)

of a growing band along paths provided by an intersecting set of bands is shown in Fig. 40. Portions of the original crystal and other twin bands are left in the interior of the growing band. This irregular growth, of course, accounts for patches of irregular shape, the so-called recrystallized grains, which according to this view possess the orientations of the original crystal or its four possible twins.

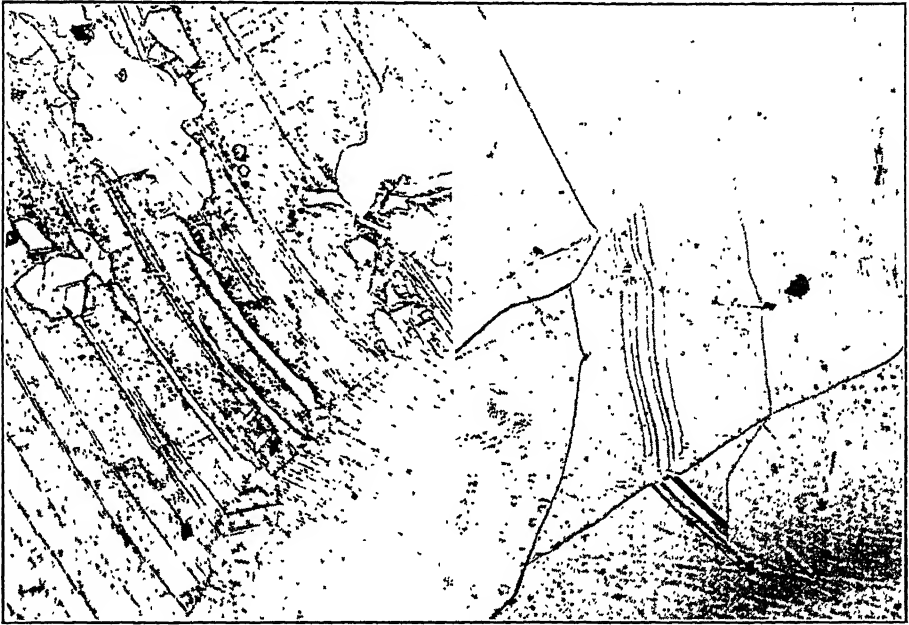


FIG. 41.—FORMATION OF CURVED TWIN BANDS. $\times 500$.

FIG. 42.—CURVED TWIN BANDS IN A FULLY ANNEALED SAMPLE OF ALPHA BRASS. $\times 100$.

(Reduced to $\frac{4}{5}$ original scale, original magnification given.)

General distortion might bring about some deviation from this simple condition in that the parts of a distorted twin band might grow into separate grains of slightly different orientation. That this need not always occur is shown by Fig. 41, in which curved bands have grown together on annealing with a probable readjustment of the atoms to form an unstrained lattice. A good illustration of this effect in a more fully annealed sample may be seen in Fig. 42.

When a crystal which has already been worked⁴⁶ and annealed once is again treated in the same manner, each visible twin band or grain emanating from a twin band is potentially capable of developing a new set of

⁴⁶ It is assumed that all forms of cold working yield twins in most face-centered cubic metals. Even though this is not at once apparent in the microstructure after cold working it is inferred from the numerous twins which are observed after annealing.

bands or grains. The twelve new orientations possible in such a second generation of twins are shown by the open squares in the stereographic projection reproduced in Fig. 43. All orientations are referred to an original plane of form $\{100\}$, the plane of the projection, and the assumption is made that there is no deviation from the reorientation produced exclusively by twinning. If this does not hold each point on the projection might be replaced by a number of points in this general vicinity. A third sequence of working and annealing would be expected to produce a third generation of twins, making a grand total of 53 from a single individual. This process might continue indefinitely.

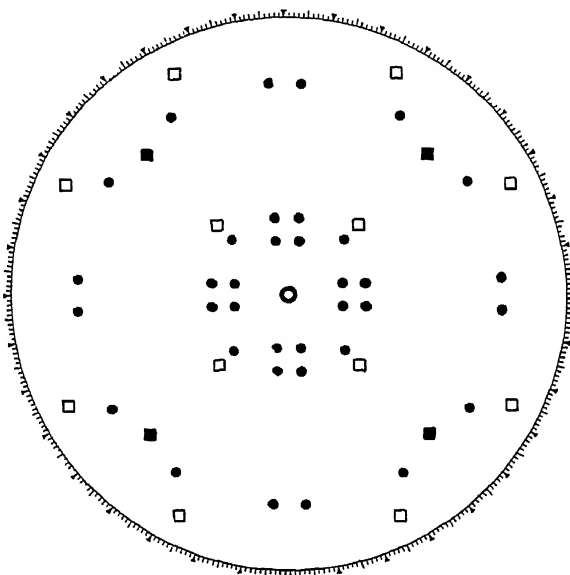


FIG. 43.—STEREOGRAPHIC PROJECTION SHOWING NEW ORIENTATIONS PRODUCED IN A FACE-CENTERED CUBIC CRYSTAL BY TWINNING ALONG ALL POSSIBLE PLANES OF FORM $\{111\}$ THROUGH THREE GENERATIONS.

- = pole of (001) in the original crystal.
- = poles of corresponding planes in twins of the first generation.
- = the poles in twins of the second generation.
- = the poles in twins of the third generation.

It is quite clear that these views give us a new point of departure for considering the effect of various mill schedules on the quality of the product. Thus, a cast material rolled to a certain gage and finished with one anneal would be less uniform and more directional in its properties than the same material annealed several times during the process, although both might look very similar under the microscope. Or, a metal such as aluminum, which apparently fails to develop twin crystals, would be less uniform than a metal such as alpha brass, which twins freely, after comparable treatment.

DISTORTION AT TWIN BOUNDARIES

A complete crystallographic description of any particular form of twinning is possible when the orientations of the crystal and its twin are known. Simple and clarifying assumptions may be made regarding the condition of the lattice in the zone of contact between crystal and twin but the impossibility of directly observing the change in position of the atoms during the operation of twinning and the lack of a method capable of revealing the crystal structure in any localized (boundary) region have left us dependent upon indirect methods of testing these assumptions.

Preston⁴⁷ finds that planes of form $\{111\}$ in the face-centered cubic metals and $\{211\}$ in body-centered cubic metals permit twinning with the least amount of distortion and the most perfect continuity of structure under the fundamental assumption that the components of the twin have in common at least one plane of atoms. Distortion in the sense that atom centers across the twinning plane come closer than the closest distance of approach of atoms in the primitive material would be entirely absent in this ideal form of face-centered cubic twinning, but would be present to a slight degree even in this most favorable form of body-centered cubic twinning.

A possible form of distortion in the twinning of zinc along planes of form $\{10\bar{1}2\}$ was shown by Mathewson and Phillips⁴⁸ and Fig. 44 represents the development of a similar plan involving an intersection between two conjugate bands. One of the planes of form $\{10\bar{1}2\}$ is maintained in each twin and the twinning is supposed to occur by progressive slip together with a slight secondary adjustment, along the neighboring planes of this form. There does not appear to be any plane that would permit twinning without distortion in hexagonal close-packed metals even when the axial ratio corresponds to close packing of spheres.

If both parts of the crystal move as a whole each time a layer is added to a simple twin band, the distortion may be confined very closely to the boundaries as shown in the original figure, but if a twin lamella merely bridges a gap between two parts of the same crystal⁴⁹ there must be a zone of distortion in which the horizontal rows of atoms at the right of the band rise to their natural level in the unaltered crystal. This condition is represented in Fig. 45. The very common tendency of twins in zinc, cadmium, and tin to taper probably represents a natural tendency to avoid abrupt transitions of structure and hence strongly localized distortions in any part of the material. An estimate of the

⁴⁷ G. D. Preston: The Formation of Twin Metallic Crystals. *Nature* (1927) 119, 600.

⁴⁸ C. H. Mathewson and A. J. Phillips: Plastic Deformation of Coarse-grained Zinc. *Proc. Inst. of Metals Div. A.I.M.E.* (1927) 186.

⁴⁹ Twins produced by local indentation, for example.

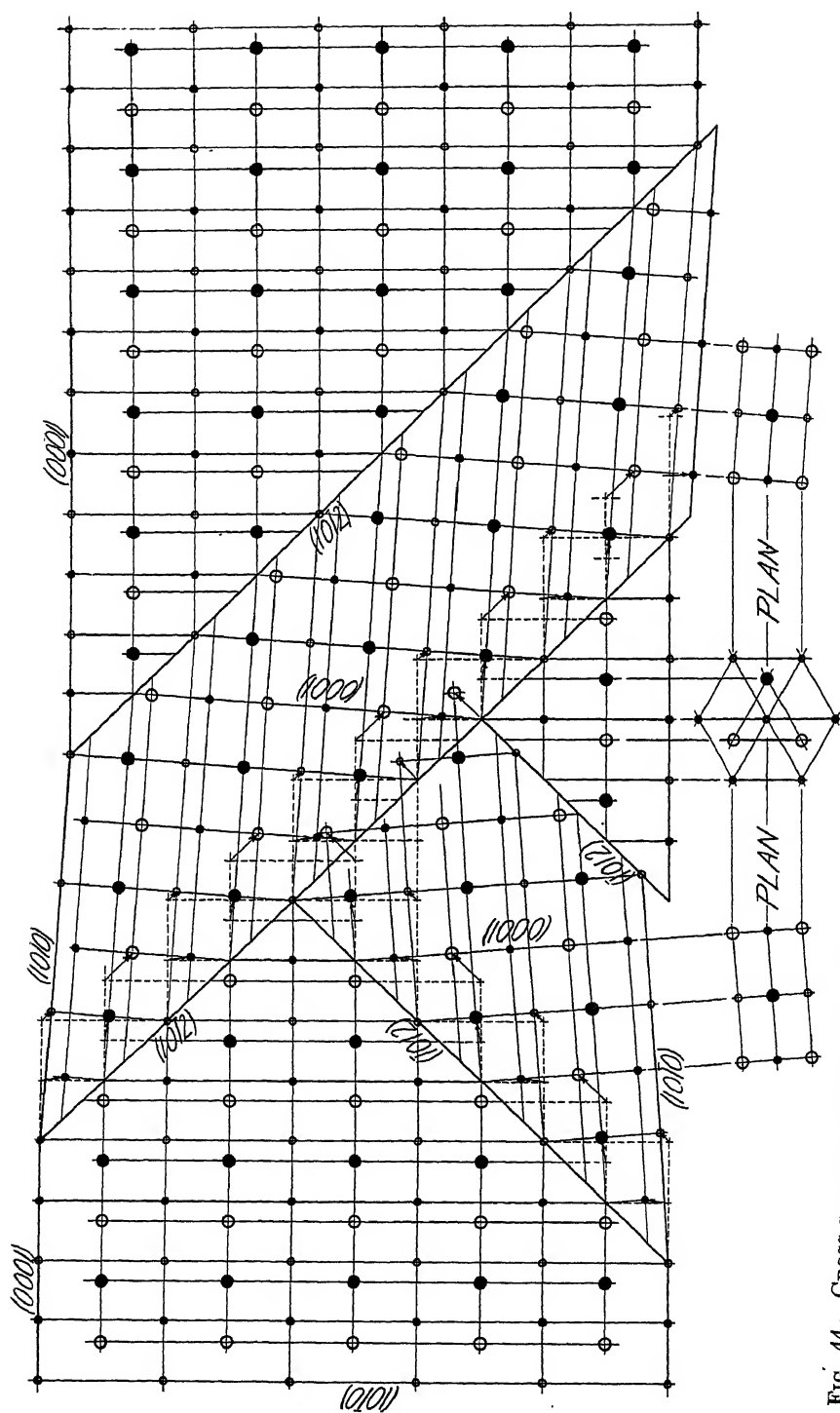


FIG. 44.—CROSS-SECTION THROUGH TWIN BANDS IN ZINC. TWINNING PLANES PERPENDICULAR TO THE PLANE OF THE SECTION. SHOWS POSSIBLE FORMS OF DISTORTION AT TWIN BOUNDARIES.

results which may be secured in this manner may be made by comparing Fig. 45 with Fig. 46.

The examples already introduced leave little room for doubt that simple twinning within a crystal may generate twin boundaries either

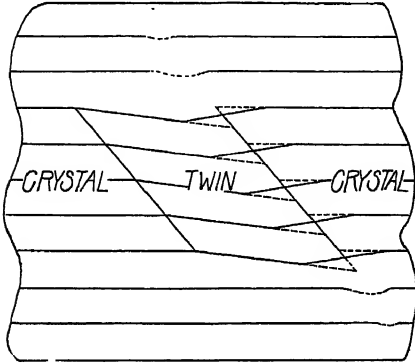


FIG. 45.—POSSIBLE ZONES OF SEVERE DISTORTION AT THE BOUNDARIES OF AN INTERNAL TWIN BAND.

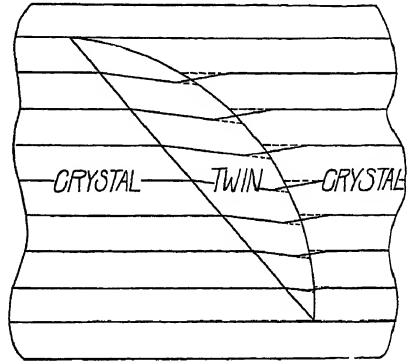


FIG. 46.—GRADUATED DISTORTION AT THE BOUNDARIES OF A TAPERED INTERNAL TWIN BAND.

free from strain or accompanied by strain varying greatly in amount and distribution according to the forms of crystal lattice and twinning plane and the configuration of the twinned area.



FIG. 47.—WORKED AND ANNEALED ZINC.

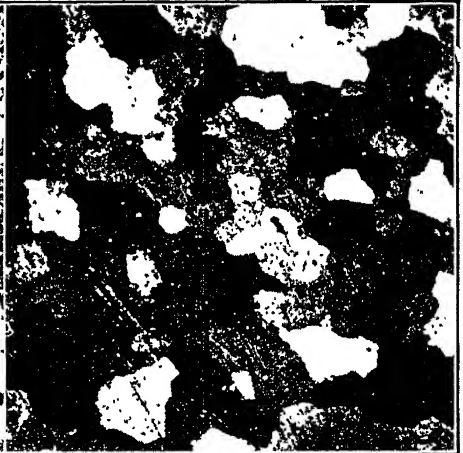


FIG. 48.—WORKED AND ANNEALED TIN.

Small twins with unstrained boundaries often grow into large ones, as is so commonly observed in the annealing of copper and its alpha solutions. I can find no evidence that such a process occurs in the case of twins with strained boundaries, although certain large twins of this character may retain their original shape on annealing, as observed by

Mathewson and Phillips in zinc⁵⁰ and McKeehan in iron.⁵¹ Commonly, small twins with strained boundaries fail to retain the banded structure and ultimately produce grains with irregularly shaped boundaries, as in Fig. 32 from Krivobok's work on iron,⁵² Fig. 47 representing worked and annealed zinc, and Fig. 48 representing worked and annealed tin. These facts give support to the fundamental assumption of a plane

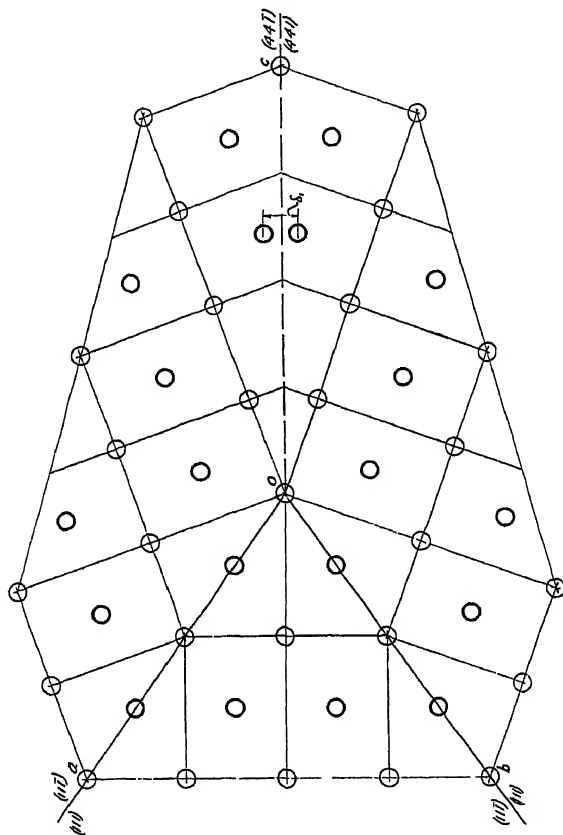


FIG. 49.—PROJECTION OF PART OF A FACE-CENTERED CUBIC CRYSTAL TWINNED ALONG TWO PLANES OF FORM $\{111\}$ WHICH MEET AT AN ANGLE OF $70^\circ 32'$. PLANE OF THE PROJECTION PERPENDICULAR TO THE LINE OF INTERSECTION OF THESE PLANES.

common to both components of a twin and the accompanying interpretation of strain.

A close connection between twinning and recrystallization in copper and alpha brass was observed in the experiments of A. J. Phillips. It was shown that the narrow indelible markings (etch bands) resembling Neumann bands in ferrite coalesce directly into twin bands of ordinary

⁵⁰ C. H. Mathewson and A. J. Phillips: Plastic Deformation of Coarse-grained Zinc. *Proc. Inst. Metals Div. A.I.M.E.* (1927) 183.

⁵¹ L. W. McKeehan: *Op. cit.*

⁵² V. N. Krivobok: *Op. cit.*

appearance. The frequent occurrence of new orientations (recrystallization) was undoubtedly due to a lack of perfect homogeneity within the bands as a whole. While no evidence could be adduced to prove the absence of distortions due to causes other than twinning, the obvious disturbance caused by the interpenetration of two sets of bands led to

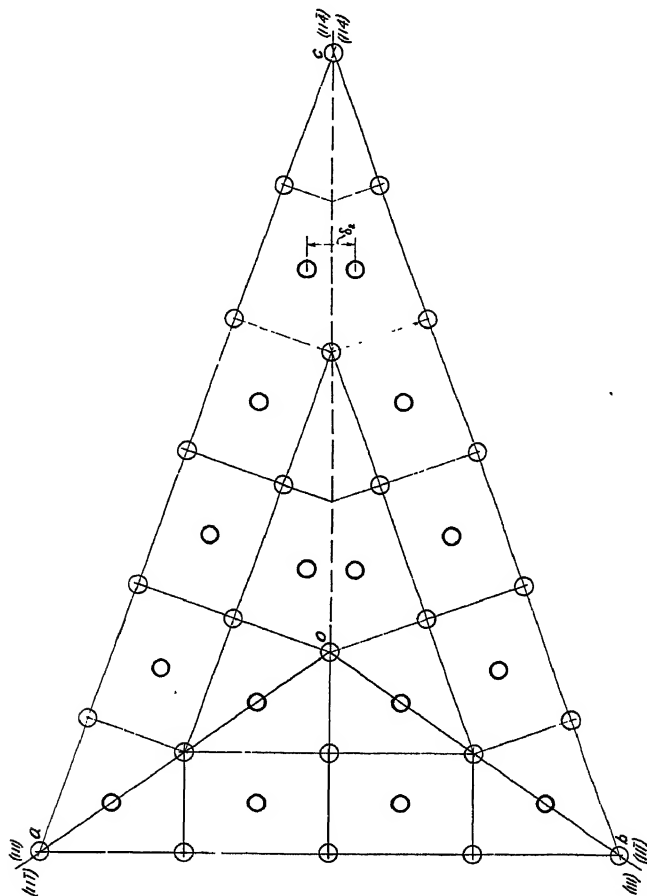


FIG. 50.—PROJECTION OF PART OF A FACE-CENTERED CUBIC CRYSTAL TWINNED ALONG TWO PLANES OF FORM $\{111\}$ WHICH MEET AT AN ANGLE OF $109^{\circ} 28'$. PLANE OF THE PROJECTION PERPENDICULAR TO THE LINE OF INTERSECTION OF THESE PLANES.

the belief that such intersections were at least a principal source of the distortion which is responsible for recrystallization.

I am indebted to my colleague, L. W. McKeehan, for the following analysis of the twin relationships in a face-centered cubic crystal.

Fig. 49 is a projection of a portion of the crystal twinned along two planes of form $\{111\}$ which make an angle of $70^{\circ} 32'$. The plane of the projection is perpendicular to the line of intersection of the two composition planes, shown at oa and ob . Since oa and ob are planes of form

$\{111\}$ their intersection is a line of form $[110]$. The plane of projection is therefore of the form $\{110\}$. The two different twins aoc and boc , which were derived from the crystal $ao b$, are mirror images of each other in the plane oc , of form $\{441\}$ and the closest distance of approach of atom centers, δ_1 , is only 0.33 of the least distance before twinning. Fig. 50 is a similar projection in which the pairs of twinning planes make an angle of $109^\circ 28'$. In this case the new composition plane, oc , between the twins becomes of the form $\{411\}$ and the closest distance of approach of atom centers is δ_2 , or 0.47 of the normal distance before twinning. Both cases, therefore, require a great amount of distortion, as was suspected from the experimental evidence.

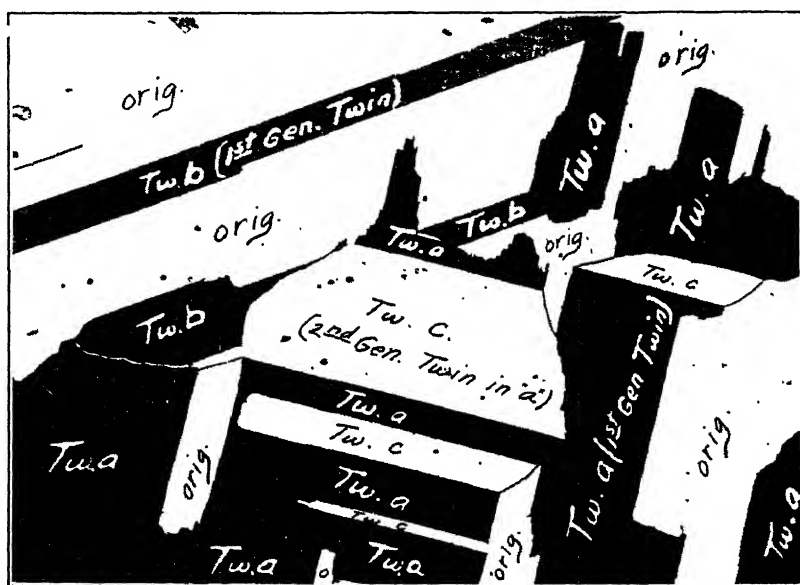


FIG. 51.—TWO GENERATIONS OF TWINS IN A SINGLE BRASS CRYSTAL.

From these considerations it is evident that the absence of distortion in the vicinity of the composition plane between a crystal and its twin does not imply a similar absence of distortion at the surfaces of contact between various twins formed primarily within the original crystal or secondarily within these twins of the first generation.

The stereographic projection given in Fig. 43 exhibits the entire group of 53 orientations which may be formed by multiple internal twinning along planes of form $\{111\}$ through three generations in a face-centered cubic crystal.

Probably many of the possible combinations embraced in this projection with great diversity in the amount of distortion at the boundaries are encountered during the succession of cold working and annealing operations to which metals are frequently subjected. That twins of at

least two generations may be found together with a residue of the original crystal in a fully annealed product is proved by the examination made by A. J. Phillips⁵³ of the crystal shown in Fig. 51. Here, *orig.* or *o* is the original crystal, *Twa* and *Twb* are twins of the first generation and *Twc* is a twin of the second generation in *Twa*.

It is believed that many and perhaps most of the new orientations formed during the recrystallization of these face-centered cubic metals are due to complex twinning resulting in strained boundaries which permit growth and readjustment in various forms.

COMPARATIVE FUNCTIONS OF TWINNING AND SLIP IN THE GENERAL PROCESS OF PLASTIC DEFORMATION

Simple twinning, unaccompanied by slip, does not provide for any considerable change in the dimensions of a crystal and cannot account for the great amount of plasticity which is observed in the ordinary operations of rolling, drawing, etc. The possibilities of twinning and slip in this respect are compared in Figs. 52 and 53. These sketches represent sections perpendicular to a twinning plane, *t*, and a slip plane, *s*, respectively. The initial thickness is represented by the vertical distance, *d*.

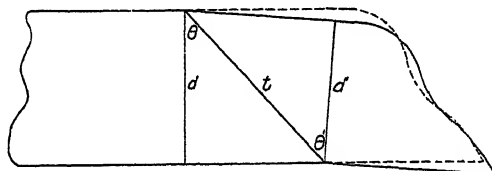


FIG. 52.—CHANGE IN THICKNESS BY TWINNING. TWINNING PLANE PERPENDICULAR TO PLANE OF SECTION.

Twinning across the entire section, as shown in Fig. 52, causes an equal angular displacement of the upper and lower surfaces of the crystal from the horizontal plane and changes the thickness to d' . There is, of course, a corresponding change in orientation. If Θ represents the angle between the twinning plane and the vertical plane and, similarly, Θ' , the angle between the twinning plane and the plane perpendicular to the crystal surfaces after twinning, the expression,

$$\frac{d}{d'} = \frac{\cos \Theta}{\cos \Theta'}$$

gives the ratio of thickness before and after twinning.

The angles Θ and Θ' can be changed only by changing the initial orientation.

In the case of simple slip, as shown in Fig. 53a, the surfaces of the crystal move out of the horizontal plane but acquire a stepped configura-

⁵³ A. J. Phillips: *Op. cit.*

tion which approaches a plane surface as the number of slip planes contributing to the total displacement becomes infinite. In reality this number is limited by the finite distance between consecutive planes of the form under consideration. The ratio of thickness before and after the deformation, as in the case of twinning, is given by the expression

$$\frac{d}{d'} = \frac{\cos \Theta}{\cos \Theta'}$$

However, the angle Θ' , in this case can be increased, as shown in Fig. 53b, by increasing the amount of slip. In this way, the thickness may be continually reduced as long as the slipping process can be made to operate.

Simple slip along one set of planes changes the plane of the surface without changing the orientation. Doubtless many movements of this

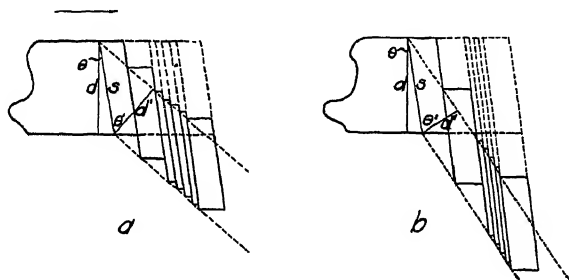


FIG. 53.—CHANGE IN THICKNESS BY SLIP. SLIP PLANE PERPENDICULAR TO PLANE OF SECTION.

sort take place during plastic deformation of an aggregate of crystals of random orientation. Unless a principal component of stress remains parallel to the moving surface as simple slip proceeds, which would appear to require some special supporting or adjusting mechanism, the crystal must yield in another direction. This may bring into play another set of slip planes or a set of twinning planes if either are available, while in the absence of such favorable conditions progressive curvature of the slip planes, or certain localized lattice distortions, must occur as the movable surface is forced continually into line with the direction of stress.

It was pointed out by Jeffries and Archer⁵⁴ that the wedge-shaped fractures observed by Sykes in 1921⁵⁵ in molybdenum wires were probably

⁵⁴ Zay Jeffries and R. S. Archer: Effect of Temperature, Pressure and Structure on Mechanical Properties of Metal. *Chem. & Met. Eng.* (1922) 27, 747.

⁵⁵ W. P. Sykes: Effect of Temperature, Deformation, Grain Size and Rate of Loading on Mechanical Properties of Metals. *Trans.* (1920) 64, 780.

produced by cooperative movements of the first kind, namely, slip along conjugate planes inclined 45° to the axis of the wire.

Later Goucher⁵⁶ in a series of three papers very carefully analyzed these wedge-shaped fractures through single crystals of tungsten and ascribed them to slip along conjugate planes of form $\{211\}$ in the direction $[111]$. Even in the case of the most symmetrical wedges, which would appear to favor slip in its simplest form, rather large amounts of distortion were observed, which in the words of the author "affected the inclination of the crystal planes in the direction of slip, but produced no marked change in the atomic spacings." This indicates that severe distortion which might be visualized as curvature of the slip planes in the above sense actually results in fragmentation, or disintegration of the crystal into small lattice units arranged on curved surfaces.

Taylor and Elam⁵⁷ working with aluminum, and later Elam⁵⁸ with other face-centered cubic metals and alloys, have described slip along single and conjugate planes of form $\{111\}$ in single crystals of these materials with characteristic changes in orientation as the slip planes tend to reach a position of equilibrium in conformity with the loading conditions adopted. No particular theory of lattice distortion was favored by these authors.

On the other hand, Polanyi, in a number of communications,⁵⁹ vigorously asserts that cold working in general produces lattice distortion by a combined process of slip and bending (*Biegegleitung*) which was first observed in zinc by Mark, Polanyi and Schmid.⁶⁰ If only one set of slip planes can operate in the case of zinc, tensile stress in the direction of the arrow in Fig. 53 must either distort the lattice (by *Biegegleitung*) or bring some form of compensatory movement into play. Mathewson and Phillips⁶¹ suggested that twinning across the slip planes might satisfy

⁵⁶ F. S. Goucher: On the Strength of Tungsten Single Crystals and Its Variation with Temperature. *Phil. Mag.* (1924) **48**, 229.

Studies on the Deformation of Tungsten Single Crystals Under Tensile Stress. *Phil. Mag.* (1924) **48**, 800.

Further Studies on the Deformation of Tungsten Single Crystals. *Phil. Mag.* (1926) **2**, 289.

⁵⁷ G. I. Taylor and C. F. Elam: The Distortion of an Aluminum Crystal during a Tensile Test. *Proc. Roy. Soc.* (1922-3) **102A**, 643; also The Plastic Extension and Fracture of Aluminum Crystals. *Proc. Roy. Soc.* (1925) **108A**, 28.

⁵⁸ C. F. Elam: Tensile Tests of an Aluminum Zinc Alloy. *Proc. Roy. Soc.* (1925) **109A**, 143; also Tensile Tests of Large Gold, Silver and Copper Crystals. *Proc. Roy. Soc.* (1926) **112A**, 289 and Tensile Tests on Alloy Crystals. *Proc. Roy. Soc.* (1927) **115A**, 133 and (1927) **116A**, 694.

⁵⁹ For instance: Kristalldeformation und Verfestigung. *Ztsch. Metall.* (1925) **17**, 94.

⁶⁰ H. Mark, M. Polanyi and E. Schmid: *Op. cit.*

⁶¹ C. H. Mathewson and A. J. Phillips: Plastic Deformation of Coarse-grained Zinc. *Proc. Inst. Metals Div., A. I. M. E.* (1927) 144.

this condition. This has recently been denied by Wilson and Hoyt,⁶² although possibly on insufficient grounds.⁶³

Whatever the relative functions or interrelationship of slip and twinning in zinc crystals under simple forms of static loading, there is no doubt that these two processes cooperate and are mutually indispensable in facilitating the plastic deformation of zinc by ordinary methods of cold working.

The first stage of rolling on a basal surface already described in some detail is especially significant in this respect. The reduction in thickness in this experiment (6.98 per cent.) corresponds almost exactly to the amount required by complete twinning along planes of form $\{10\bar{1}2\}$ and very large portions of the specimen are found entirely converted to the twin orientation. This corresponds to the condition shown in Fig. 52 requiring the twinned part of the crystal to develop surface planes inclined to the original surface and at once suggests that a forward portion of the specimen tilts downward as it twins between the rolls and thereupon emerges

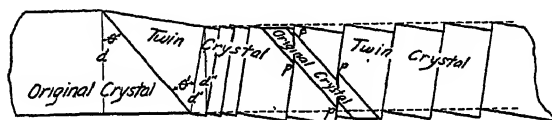


FIG. 54.—PYRAMIDAL TWINNING IN A ZINC CRYSTAL FOLLOWED BY BASAL SLIP IN TWIN LAMELLAE. TWINNING AND SLIP PLANES PERPENDICULAR TO PLANE OF SECTION.

at the proper angle to the material just entering the rolls. Other parts of the surface, however, have undoubtedly left the rolls in a horizontal plane in spite of twinning, a condition which seems to require slip according to some plan similar to the one shown⁶⁴ in Fig. 54. The thickness measured in the vertical plane after twinning along $(01\bar{1}2)$ planes and basal slip in the twinned material would be given by the expression, $d'' = d \frac{\cos \theta'}{\cos \theta} \cos (\theta' - \theta)$ in which $\theta = 42^\circ, 56'$, $\theta' = 47^\circ, 4'$ and d , the initial thickness, is taken as unity. This represents a reduction in the thickness of 7.21 per cent., which is a little more than the amount attained in the experiment.

The possibility of prismatic slip through residual lamellae of the original crystal is illustrated at p in the figure. This would introduce transverse strain, which might account for some of the twinning across such bands. Actual observations in rolling show that twinning does not

⁶² T. A. Wilson and S. L. Hoyt: X-ray Analysis of Plastic Deformation of Zinc. *Proc. Inst. Metals Div., A. I. M. E.* (1928) 241.

⁶³ Cf. discussion of Wilson and Hoyt's paper by L. W. McKeehan.

⁶⁴ In order to simplify the discussion it is assumed that the twinning planes are perpendicular to the exposed vertical section representing the direction of rolling in these experiments.

follow a single set of planes all the way through the specimen, as in Fig. 54, but is distributed among two sets of conjugate planes, closely parallel to the principal shear components of stress, one of them brought into play by the action of the upper roll and the other by the lower roll. This condition is illustrated in Fig. 55. When the end of the specimen gets the full pinch of the rolls it is twinned from top to bottom as represented in the shaded area of Fig. 55a. On further progress through the rolls twinning might spread out along the conjugate planes (01 $\bar{1}$ 2) and (0 $\bar{1}$ 12) without causing one triangular zone to invade the other, by pushing the end blocks along the (shaded) slip planes of form {0001} to generate new horizontal surfaces as shown in Fig. 55b.

The symmetrical effect shown in these two sketches has not been observed. Irregularities, such as eccentric loading or variation in composition, soundness or crystalline perfection of the metal commonly alter the distribution of stress and give rise to variously complicated structures of the type shown in Fig. 55c.

In case the thickness of the crystal is not reduced the full 6.98 per cent.,⁶⁵ which corresponds to complete twinning from top to bottom, alternate lamellae of the original crystal and its twin will be found as in Fig. 55d. The conditions shown in Figs. 55c and 55d may be combined to yield the highly complicated condition shown in Fig. 55e. This is quite representative of the structures actually observed.

As the crystal rolls out, twinning continues to feature the microstructure of the product and there is clear evidence of slip on the polished sides of a specimen. It seems probable that both twinning and slip are vitally concerned in the process, the former offering a rational plan for a shifting of the orientation within the mass to meet the stress conditions encountered from time to time and the latter providing for most of the plastic thinning of the crystal.

It is possible to imagine a unique combination of twinning and slip which would permit a succession of transformations from original crystal to twin and back to original crystal, with a reduction in thickness at each stage of the process. The central section of Fig. 56 shows the positions of the atoms in a twinned layer of a zinc crystal with twinning plane perpendicular to the plane of the section. The change from original position to twin position of all atoms in the first four rows parallel to the row held in common by the crystal and its twin is indicated by arrows. For comparison a similar set of five rows free from these construction details is shown by dotted lines a short distance to the right. The five rows constitute a block bounded by two geometrically similar planes and the action in this block may be considered representative of the process as a whole. It is believed that incipient or fractional slip

$$\frac{65 \frac{1.862 - \sqrt{3}}{1.862}}{1.862} \times 100.$$

in these rows brings the atoms near enough to the positions they would occupy in the twin to determine an automatic and instantaneous completion of the twinning process.

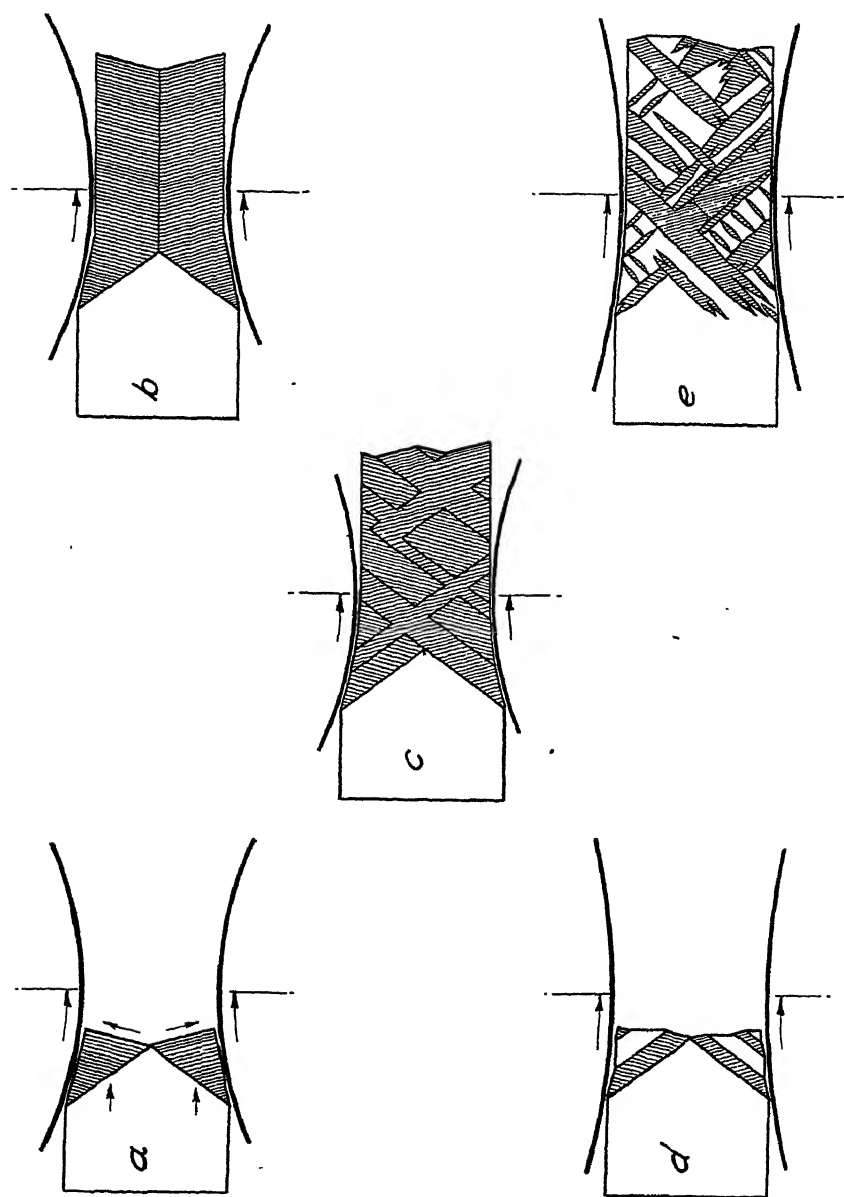


FIG. 55.—TWINNING AND SLIP IN A ZINC CRYSTAL BY ROLLING. SECTION PERPENDICULAR TO (0001) AND (0112). FULL TWINNING REDUCTION IN *a*, *b*, AND *c*; PARTIAL REDUCTION IN *d* AND *e*.

The general effect of this change is to deform the block by lowering its forward boundary (the fifth row of atoms) an amount corresponding to the fractional slip shown just below the first figure 5 in the sketch and adjusting the positions of the atoms in the interior.

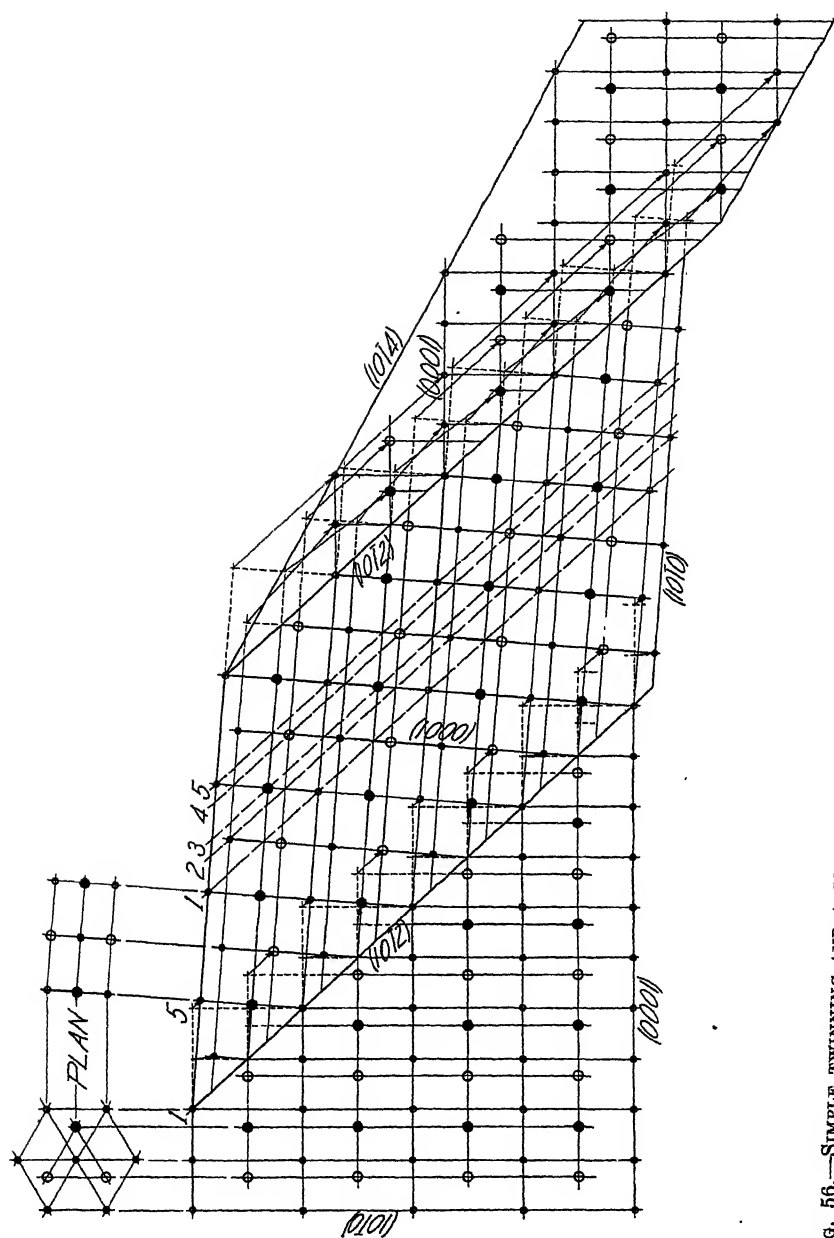


Fig. 56.—SIMPLE TWINNING AND A CONJECTURED RETURN TO THE ORIGINAL ORIENTATION WITH PROGRESSIVE REDUCTION IN THICKNESS. TWINNING PLANES PERPENDICULAR TO PLANE OF SECTION.

If now, we deform a similar block in the twin by lowering its forward wall far enough to complete a fractional slip of the above character and suitably readjust the atoms in the interior, the orientation of the original crystal will result with a very considerable reduction in thickness as shown in the right-hand section of the sketch.

The first part of the process just described seems probable as it moves the atoms much less than would correspond to a full slip in any case and is quite in harmony with the recorded observations on twinning. On the other hand, there is no evidence that the last part of the process ever occurs. It is perhaps improbable on account of the very considerable relative displacements of the atoms.

We have seen that twinning is capable of introducing strain and thereby supplying the incentive for recrystallization. From a crystallographic point of view this form of lattice reconstruction is more systematic and less arbitrary and quite likely of more frequent occurrence than the other alterations in crystal structure involving distortion, such as rotation during slip around an axis perpendicular to the slip plane or around an axis lying in the slip plane (*Biegegleitung*), which have been observed under special conditions and are supposed to occur freely in the case of severe deformation.

POSTULATES CONCERNING TWINNING IN RELATION TO THE PLASTIC DEFORMATION OF METALS

1. When in the inhomogeneous deformation of a metal by slip combined with distortion, the atoms in certain zones reach positions more nearly approximating equilibrium positions in a possible twin than in the original crystal, twin lamellae will be formed with partial relief of stress and the residual distortion will be distributed in harmony with the geometrical requirements of each special case.

2. By far the great majority of the metals are subject to structural alteration of this character.

3. Under slowly applied loads there is a minimum occurrence of the continuous distortion from plane to plane required to initiate twinning, owing to the tendency toward premature slip along "weak" planes of variable spacing depending upon the concentration of impurities or the operation of other factors which may affect the cohesion between planes.

4. Under impact loading a given small displacement is momentarily distributed among many planes, thus disturbing the equilibrium of large numbers of atoms and potentially favoring the formation of sizable twin bands.

5. The distortion around simple twin bands wholly enclosed within the original crystal may be confined to the ends as distinguished from the parallel sides bounded by twinning planes and this brings about end

growth on annealing which may spread along the sides. In other cases distortion may exist at the composition planes and bring about direct lateral growth.

6. Both the quantity and the size of deformational twins are greatly affected by the kind of deformation and the purity of the metal.

7. Distortion and thermal instability are especially pronounced in the vicinity of faults or direct intersections where twin bands enter into conflict with one another.

8. One form of recrystallization is essentially granulation produced by irregular and competitive growth of interstratified twin lamellae which are here termed crystallographic fragments in that they cannot be fitted together without distortion in spite of their crystallographic relationship through the parent crystal.

9. Recrystallization in the absence of twinning is a similar reaction between lamellae (fragments) formed by inhomogeneous distortion during slip, *e. g.*, variable rotation of the slip planes. This leads to less pronounced changes in orientation from grain to grain than is effected by twinning.

10. In conformity with the conception of recrystallization expressed in (8) and (9), diversity of orientation after a succession of deformational and annealing treatments is due to multiple internal twinning, rotation during slip or some combination of these two causes depending upon the nature of the metal, the initial orientation of the crystals, and the kind of deformation.

Purification of the Six Platinum Metals*

BY EDWARD WICHERS,† RALEIGH GILCHRIST‡, AND WM. H. SWANGER,§

WASHINGTON, D. C.

(New York Meeting, February, 1928)

THE purpose of this paper is to set forth the matters of principal interest in connection with work done in the past few years on the purification of the metals of the platinum group, including the study and modification of methods already in the literature and the development of new procedures. For the most part, the results will be given in summary form, reserving the publication of detailed data and directions for a forthcoming paper to be issued by the Bureau of Standards.

In recent years increasing interest has been shown in the preparation of the pure platinum metals, both by those concerned with the manufacture of the metals and their alloys for scientific and industrial purposes and those who are engaged in the study of the chemical or physical properties of the elements. The purification of the metals usually has been regarded as a matter so involved and difficult as to require the attention of specialists. It is hoped that this paper will show that any of the metals can be prepared in any desired degree of purity by relatively simple straightforward chemical methods.

For the most part, it is thought that the methods given are suitable for small-scale refineries as well as for the laboratory; also, that they will bring the time required for the purification of any of the metals within reasonable limits. The new procedure suggested for purifying platinum should prove to be much shorter than any methods described in the literature.

TREATMENT OF PLATINUM ORE

As is well known, the principal source of the platinum metals is a mixture of several native alloys of the metals, found chiefly in naturally concentrated deposits in stream beds or alluvial material. Without attempting to enumerate all of the minerals present, it may be said that the concentrated ore from such sources contains all of the six platinum

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metals, together with gold, silver, copper, iron, and possibly other base metals, in a number of alloys, as well as small amounts of a number of nonprecious minerals which are not completely separated in the process of concentration. A few compounds of the metals, such as platinum arsenide and ruthenium sulfide, are known. In certain of the natural occurrences one or more of the metals may be practically absent, as in the case of some osmiridium deposits, which contain little or no platinum or palladium. Similarly, little if any iridium, osmium, or ruthenium is found with the platinum and palladium associated with copper and nickel ores. Platinum, however, is the most abundant of the six metals and predominates in most of the ores.

A scheme of separation suitable for an average platinum ore, like those produced in Russia or Colombia, will be found adaptable, with appropriate modifications, to almost any type of concentrated platiniferous material. Such a scheme is described first. In the subsequent sections of the paper, dealing with the individual metals, it will be understood that the original material contains relatively small proportions of the remaining platinum metals or other elements.

In the usual practice, the ore is first digested with aqua regia, fresh acid being added from time to time as long as any appreciable amount of metal dissolves. A mixture of four volumes of hydrochloric acid (sp. gr. 1.18), one volume of nitric acid (sp. gr. 1.42), and one volume of water is satisfactory for the purpose. The resulting solution contains the bulk of the platinum and palladium as well as the gold and most of the base metals present in the material. It will also contain iridium, rhodium, and small amounts of ruthenium which were present as minor constituents of the soluble alloys, but any osmium which dissolves escapes as the volatile tetroxide. Some or all of the silver will be in the solution because of the solubility of silver chloride in such a strongly acid medium. Osmiridium and other alloys containing iridium as a major constituent remain in the insoluble residue, together with most of the ruthenium, much of the rhodium and much of the nonprecious mineral content of the sample. Some platinum and presumably palladium are likely to be present in the insoluble portion.

GENERAL SEPARATIONS

Treatment of the Aqua Regia Solution

The first operation is to precipitate the bulk of the platinum with ammonium chloride. To prepare the solution for this precipitation the nitric acid and most of the excess of hydrochloric acid are removed by evaporation. A small amount of water is added to the hot sirupy residue to cause the decomposition of nitrous compounds of platinum, after which hydrochloric acid is added and the evaporation repeated. It

is well to continue the evaporation each time until the temperature of the solution reaches 140° to 150° C., unless the residue becomes pasty, in which case local overheating is likely to occur. The evaporation at high temperature probably causes the reduction of part of the iridium to the trivalent state, in which form less of this element will be precipitated with the platinum salt.

After the solution has been evaporated twice it is diluted so as to contain 50 to 100 g. of platinum per liter and set aside to permit silver chloride and other insoluble matter to settle. The solution is decanted or filtered from the insoluble residue, heated nearly to boiling and treated with ammonium chloride. Ammonium chloroplatinate separates at once. A small excess of ammonium chloride is desirable to reduce the solubility of the ammonium chloroplatinate, but a large excess should be avoided because it increases the degree of contamination of the salt. The solution is cooled quickly and filtered at once to avoid contaminating the precipitate with the very impure salt which separates on standing. The precipitate is well drained and washed once or twice with a 20 per cent. solution of ammonium chloride, after which it is dried and ignited to sponge. The filtrate contains varying amounts of platinum but usually less than 5 per cent. of the platinum in the sample.

The next step depends on the nature and proportion of the other metals present. If there is a large amount of gold, it may be desirable to separate that metal by the addition of ferrous sulfate solution. The separation is rapid and complete, and the gold thus obtained is fairly pure. If much palladium is present, it may be desirable to add more ammonium chloride and a volume of nitric acid amounting to 20 or 30 per cent. of the solution. Digestion on the steam bath causes the precipitation of ammonium chloropalladate, accompanied by much of the iridium and platinum in the solution. However, the presence of nitric acid interferes greatly with the removal of the remaining precious metals. Usually the most convenient procedure is to precipitate all of the metals remaining in the solution, after the first precipitation of platinum, by means of zinc or iron. As it is very difficult to precipitate iridium completely in this way, the process must be continued for a long time, possibly for some days. Even then it is quite likely that some of the iridium will escape precipitation.

The precipitated metals are washed by decantation and after a sufficiently large amount of such material is accumulated it is attacked with aqua regia diluted with four volumes of water. Gold and palladium are very rapidly dissolved and platinum reasonably so. Comparatively little iridium, rhodium, and ruthenium will dissolve. If platinum predominates in the solution, it is best precipitated first, by means of ammonium chloride, as described. If not already separated, gold is then removed by precipitation with ferrous sulfate, after which palladium is precipitated

as described. The various precipitates, platinum, gold and palladium, will contain more or less iridium, rhodium and ruthenium. The remainder of these metals together with unprecipitated platinum and palladium are recovered by precipitation with zinc as before, after evaporating the solution to eliminate most of the nitric acid.

Residue after Treatment of Aqua Regia Solution

The residue of rhodium iridium and ruthenium may be worked up with the osmium-iridium material which remained undissolved in the original aqua regia treatment. However, if sufficient material is at hand to work the two lots separately it will be better to do so, because the former contains little or no osmium and a very minor amount of ruthenium and may therefore be handled somewhat more simply. If rhodium is predominant, the dried residue is intimately mixed with about $2\frac{1}{2}$ times its weight of sodium chloride and brought to dull red heat in a current of chlorine. This treatment converts a large part of the rhodium to the soluble sodium rhodium chloride. Some of the iridium will also be converted to a double chloride, but this metal is less readily attacked than rhodium.

If iridium is predominant the residue is better attacked by fusion, at 600° to 700° C., with three parts of sodium hydroxide and one part of sodium peroxide in a silver, nickel or iron dish. A portion of the iridium is dissolved, probably in the form of a basic iridate. By far the greater part of the iridium from such a fusion remains insoluble in water but does dissolve in hot concentrated hydrochloric acid to form iridium chloride. The ruthenium present will be found largely in the aqueous solution obtained by leaching the fused alkaline mass. If only small amounts of ruthenium are involved it will be less tedious not to attempt to separate it at this point, but merely to concentrate it by precipitation together with iridium and any osmium by exactly neutralizing the alkaline solution with hydrochloric acid or sulfuric acid, adding a little alcohol and boiling. The small amount of metals remaining in the solution after this treatment may be recovered by reduction with zinc and hydrochloric acid. The treatment of this mixture of iridium, ruthenium and osmium is discussed in the section dealing with the osmiridium fraction. Rhodium is not rapidly attacked by alkaline fusions. For this reason it is often found advantageous to alternate the treatments, sodium chloride + chlorine and sodium hydroxide + sodium peroxide, until all the material has been rendered soluble.

When the material that consists largely of rhodium and iridium has been brought into solution as chlorides, the two metals may be separated by either of two procedures, depending on the relative proportion of the two. Unless there is much more rhodium than iridium, a convenient way of effecting the first separation is to precipitate iridium as ammonium

chloroiridate. Before adding ammonium chloride the solution should be treated with a current of chlorine to oxidize any trivalent iridium to the quadrivalent state. The solution should be concentrated so as to contain not less than 50 g. of the two metals in one liter. Enough ammonium chloride is added to react with the iridium present. A large excess should be avoided because of its interference with the subsequent concentration of rhodium. The ammonium chloroiridate is separated by filtration, drained, and washed moderately well with a solution of ammonium chloride. It is likely to contain considerable rhodium as impurity. Most of the iridium remaining in the solution with the rhodium is recovered by evaporating the solution to dryness. This serves also to eliminate any excess of acid, which would be troublesome in the subsequent concentration of rhodium. The residue is taken up in sufficient water to dissolve any readily soluble salts, such as ammonium chloride, sodium chloride, or the double chlorides of rhodium, and filtered from the small precipitate of impure ammonium chloroiridate. The filtrate is diluted so as to contain not more than 40 to 50 g. of rhodium in one liter, heated nearly to boiling and treated with sodium nitrite. This reagent first neutralizes the acid present and reacts with ammonium chloride to form ammonium nitrite, which decomposes in the hot solution. Rhodium and the other platinum metals, as well as certain base metals, are converted to soluble double nitrites, while other base metals, notably iron and tin, are precipitated as hydroxides. Heating is continued and more sodium nitrite added until the color of the solution becomes yellow or light brown. The precipitate is filtered off and treated for the recovery of the small amounts of platinum metals which it may contain. Ammonium chloride is added to the well cooled filtrate to precipitate ammonium rhodium nitrite. The details of this treatment are described in the section on rhodium. The granular, white or yellowish salt is separated by filtration, washed with water and dried, or dissolved in hydrochloric acid for further purification. It is not suitable for direct ignition to sponge. Residual metals are recovered from the filtrate by means of hydrogen sulfide.

Treatment of the Fraction Insoluble in Aqua Regia

The residue which remained undissolved in the original aqua regia treatment of the ore is converted into soluble form by fusion with sodium hydroxide and peroxide. As the grains of osmiridium are rather slowly attacked by the alkaline fusion, they are sometimes converted to a finely divided zinc alloy by melting with 5 to 10 parts of zinc at 600° to 800° C. for 2 to 3 hr. The molten mass is covered with fused zinc chloride to prevent rapid oxidation of the zinc and is occasionally stirred with a graphite rod. Treatment of the cooled melt with hydrochloric acid dissolves the excess of zinc and leaves a finely divided residue of osmium,

iridium, etc., in the form of zinc alloys. The powder is washed and dried, but not ignited, and is then ready for the alkaline fusion. The aqueous extract from the fusion with sodium hydroxide and peroxide contains practically all of the osmium and a large part of the ruthenium as well as minor amounts of iridium. The treatment of the residue that is not dissolved by water is the same as that described in the section on iridium. The alkaline solution of osmium and ruthenium is transferred to a suitable distilling flask, described in the section on ruthenium. It is strongly acidified with nitric acid, then gradually heated to boiling, a current of air being used to carry the vapors of osmium tetroxide into the chain of receiving flasks. These flasks contain a 10 to 12 per cent. solution of sodium hydroxide. A little alcohol is added to all but the first one.¹

When no more osmium tetroxide distills, as may be observed by putting a fresh solution in the first receiving flask, the contents of the several flasks are combined and digested to insure the reduction of all osmium tetroxide to sodium osmate. More alcohol is added if needed. Practically all of the osmium may be separated from the ruthenium in this distillation. It has been observed, however, that it is not possible in this way to effect a complete separation of osmium from solutions to which alcohol has been added.

After cooling the solution in the distilling flask, it is made strongly alkaline with sodium hydroxide. The solution is then saturated with chlorine, thereby converting ruthenium to the tetroxide, which distills readily when the temperature is raised to 80° or 90° C. The current of chlorine is continued during the distillation but is greatly diluted with air. The receiving flasks in this case contain hydrochloric acid diluted with four volumes of water. A little alcohol is added to all of the receivers except the first one. When the quantity of ruthenium distilling over decreases, the solution is boiled gently and the distillation continued as long as oily drops of ruthenium tetroxide appear in the delivery tube. Without cooling the solution, more sodium hydroxide is added and saturated with chlorine, whereupon more ruthenium is distilled. When very little ruthenium tetroxide is obtained on repeated distillation, the remaining iridium, ruthenium, etc., are precipitated by adding a little alcohol to the neutralized solution from the distilling flask and boiling. The contents of the receiving flasks are combined and digested to complete the reduction of ruthenium tetroxide, more alcohol being added if needed. The resulting solution is evaporated and set aside for further purification, or ruthenium may be precipitated with ammonium chloride, as described in the section dealing with that metal.

¹ A solution of ammonium hydroxide and ammonium sulfide may be substituted for sodium hydroxide. In this case the whole solution is evaporated to dryness and the residue ignited under hydrogen to osmium sponge. Some sulfur is retained by the metal.

PLATINUM

Precipitation as Ammonium Chloroplatinate

Of the methods that have been used for the purification of platinum, repeated precipitation of ammonium chloroplatinate undoubtedly is the most important. The salt is relatively insoluble. It may be readily precipitated in a form that is easily filtered and washed, and it may be directly converted to metallic platinum by ignition. Any desired degree of purity may be attained by a sufficient number of reprecipitations. However, the number of precipitations required to produce very pure metal such as that used for thermocouples and resistance thermometers is somewhat surprising. It might be expected that in this, as in other processes of purification by recrystallization, the principal cause of difficulty would be the formation of other salts which are isomorphous with that of platinum, especially if they are of the same order of solubility. It is true that all of the other metals except rhodium can exist in the quadrivalent state and in this condition form relatively insoluble salts analogous to ammonium chloroplatinate and isomorphous with it. However, because of the usual course of previous separations, osmium and ruthenium are seldom present with platinum except in very small amounts. Palladium is readily reduced from the quadrivalent state to the bivalent state by heating the solution. It is probably true that iridium also is reduced in part, at least, from the quadrivalent state to the trivalent state by heating, especially if the temperature reaches 140° to 150° C. The double chlorides of bivalent palladium and trivalent iridium with ammonium chloride are much more soluble than ammonium chloroplatinate and not isomorphous with it.

With the possible exception of iridium, therefore, one might expect to eliminate the metals of the platinum group about as rapidly as gold and the base metals, by the repeated precipitation of ammonium chloroplatinate. This is not the case. Rhodium, iridium, and palladium display remarkable persistence in contaminating the platinum salt and rhodium is about as tenacious in this respect as iridium. In one preparation of pure platinum in this laboratory the original metal contained about 0.7 per cent. of rhodium and a much smaller amount of iridium. The platinum resulting from the first precipitation with ammonium chloride contained about 0.2 per cent. of rhodium and that from the third about 0.04 per cent., estimated by thermoelectric test. After seven precipitations a minute amount of rhodium could still be detected in the mother liquor from ammonium chloroplatinate corresponding to about 450 g. of platinum. Iridium was not detected after the fifth precipitation. This persistent contamination, presumably caused by adsorption, was well

recognized by such early workers as Deville, Debray and Stas,² but seems to have been overlooked in more recently published work, especially in that dealing with methods of analysis.

Silver may persist through several steps of the purification because of the solubility of silver chloride in strong chloride solutions. Part of the dissolved silver chloride is carried down with the precipitate of ammonium chloroplatinate only to redissolve with the platinum when the sponge is treated with aqua regia. However, experience indicates that silver and gold as well as base metals are usually eliminated before the last traces of platinum metals disappear.

The degree of contamination of ammonium chloroplatinate undoubtedly varies with the method of precipitation. In this laboratory no tests have been made of the purity of the salts obtained under varying conditions of precipitation, the method employed having been chosen primarily for convenience and speed in handling the salt. In general the procedure has been as follows:

The crude sponge is dissolved in aqua regia in a porcelain dish, provided with a cover glass to prevent loss by spraying. For 100 g. of metal 300 to 350 ml. of hydrochloric acid (sp. gr. 1.18), 75 to 100 ml. of water and 60 to 70 ml. of nitric acid (sp. gr. 1.42) are used. The temperature is raised rather slowly to avoid too vigorous a reaction. In some cases more aqua regia is required for complete solution.³ The solution is evaporated fairly rapidly without previous filtration until the temperature reaches 140° to 150° C. unless the residue becomes pasty before this temperature is reached, in which case the evaporation must be stopped to avoid local overheating. If no large amount of base metal or alkali salt impurities were present in the sponge the solution will still be fluid at 150°. A small amount of water is then added, which causes a lively boiling and the evolution of nitrous fumes. In this process any nitrous compounds of platinum are decomposed. Duparc and Tikonowitch⁴ state that if the nitrous compounds formed during the treatment with aqua regia are not decomposed, a considerable amount of platinum will escape precipitation with ammonium chloride. More water is added to cool the solution below 100°. After digesting for a few minutes some hydrochloric acid is added and the evaporation repeated.

² For a detailed discussion of the observations of these workers see *Procès Verbaux*, Comité International des Poids et Mesures (1877) 154, 161.

³ Platinum sponge may also be dissolved by making it the anode in the electrolysis of strong hydrochloric acid. This method is employed by at least one laboratory which refines several thousand ounces of platinum annually. The method was first used by H. C. P. Weber: Preparation of Chloroplatinic Acid by Electrolysis of Platinum Black, U. S. Bur. Standards *Sci. Paper* 82 (1907) 365.

⁴ L. Duparc and M. N. Tikonowitch: *Le Platine et les Gîtes Platinifères de l'Oural et du Monde*, 508. "Sonor" Pub. Co., 1920, Geneva.

This whole process can be repeated three or four times in a short while, insuring the elimination of nitrous compounds and presumably promoting the reduction of quadrivalent iridium to the trivalent state. After the last evaporation water only is added and the solution diluted so as to contain not less than 50 and not more than 100 g. of platinum in one liter. It is set aside to allow settling of insoluble matter and then decanted or filtered from any residue. The residue may contain undissolved iridium or rhodium as well as silver chloride, silica, and other insoluble matter.

The solution is heated nearly to boiling and treated with a 20 per cent. solution of ammonium chloride, using 55 to 60 g. of the salt for each 100 g. of platinum and adding enough in excess so that the whole solution will contain 3 to 5 per cent. of the salt. A moderate excess of ammonium chloride is desirable to decrease the solubility of the platinum salt but a large excess increases the degree of contamination too greatly. The solution is cooled rapidly and the salt immediately filtered off and drained by suction. If the solution is allowed to stand a small amount of salt of much lower purity will separate and thus contaminate the main precipitate. The salt is well drained, returned to the dish and thoroughly mixed with a wash solution containing 20 per cent. of ammonium chloride. After draining the salt, this whole process is repeated once more. The filtrates and washings are evaporated to recover most of the residual platinum in a second crop of ammonium chloroplatinate, less pure than the first, and for the recovery of other precious metals by precipitation with zinc. The salt is dried and ignited to sponge.

In dealing with very pure platinum, suitable precautions must be taken with the containers and the manner of heating to avoid contamination of the metal during ignition of the salt. Some refiners have experienced difficulty with the use of clay crucibles. The sponge thus prepared is likely to be badly contaminated with iron. Some refiners use platinum pans. In this laboratory, ordinary porcelain ware has often been used as well as some special porcelain with a glaze of high softening point. Silica dishes are probably satisfactory. Metal vapors from windings in electrically heated furnaces are to be avoided if very pure sponge is desired. This can be done by using a glazed muffle or other glazed lining between the heating element and the vessel containing the sponge. When possible it is desirable to bring about the decomposition of the salt in a reducing atmosphere, preferably hydrogen. If ammonium chloroplatinate is ignited in the air, there is always some loss of platinum, probably caused by the volatilization of platinous chloride. The amount of this loss and the mechanical loss caused by the particles of sponge carried away in the vapors vary with the rate of heating and may be considerable. After the salt is decomposed, the ignition is continued in air. If it is desired simply to prepare the sponge for another step in the process of purification

it is not necessary to exceed a temperature of 500° to 600° C. A higher temperature causes the sponge to shrink, which makes it more suitable for melting.

Collective Precipitation of Impurities

Although reprecipitation with ammonium chloride provides an entirely satisfactory way of preparing platinum in any degree of purity, the method is tedious and time consuming. Attempts have been made by various workers to devise alternative or supplementary procedures which would shorten the process of refining. Most of this work has aimed at the removal of impurities, usually by collective precipitation of the base metals and other platinum metals with alkaline reagents. At least one laboratory which refines a considerable quantity of platinum annually is making successful use of a procedure of this kind.

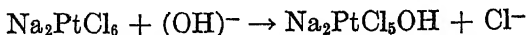
Apparently the first mention of such a scheme is in the work of Döbereiner and Weiss,⁵ who added milk of lime or lime water to the solution of impure platinum. This procedure was followed for a time at least at the Mint at St. Petersburg. Others pointed out that the precipitation of impurities was not complete and that varying amounts of platinum were present in the precipitate. Mylius and Mazzucchelli⁶ suggested the addition of a solution of sodium bicarbonate in bromine water—in effect sodium hypobromite—to a neutral solution of sodium chloroplatinate, and digesting for a short time on the steam bath, as a means of collectively precipitating most of the impurities in commercial platinum. Their scheme was used in an approximate analytical method and a complete separation was not claimed.

Work done in this laboratory has shown that under favorable conditions platinum of remarkably high purity can be obtained by the precipitation of impurities with certain alkaline reagents. It has been found that the reaction of some if not most of the platinum metal chlorides with alkaline reagents, that is with hydroxyl ions, is far from instantaneous. In order to bring to completion in a reasonably short time the reactions resulting in the precipitation of hydroxides or basic salts of these metals, it is necessary to raise the temperature of the solution to the boiling point for a few minutes. It also appears to be necessary to establish a pH of 7 to 8 or perhaps slightly beyond, that is, to have the solution very slightly alkaline and to keep it so until the precipitate has been removed, in order to insure the greatest possible elimination of impurities. This alkaline condition would not be difficult to establish and maintain except for the interference of the reaction of the chloroplatinate radical with hydroxyl

⁵ *Annalen der Pharmacie* (1835) **14**, 17.

⁶ F. Mylius and A. Mazzucchelli: *Über die Platinanalyse. Ztsch. f. anorg. Chem.* (1914) **89**, 1.

ions. This reaction is much slower than the similar reactions of the other metals but, of course, increases in speed with increase in temperature and with increase in concentration of platinum and of hydroxyl ions. It is apparent that several reactions may occur in succession, the first one being



and the others resulting in the replacement of successive chlorine atoms in the chloroplatinate radical by hydroxyl groups. The first reaction appears to produce a soluble compound, so that no difficulty is experienced with excessive precipitation of platinum. However, the pH at which this reaction comes to equilibrium in solutions having the concentration of platinum which must be dealt with in refining is distinctly on the acid side of neutrality. In other words, when a hot chloroplatinate solution is being treated with an alkaline solution to precipitate impurities as hydroxides or basic salts, platinum also tends to react with the alkali. This reaction very quickly causes the solution to become sufficiently acid to prevent the complete precipitation of the impurities or to redissolve some of the compounds already precipitated. It has been found that the presence of bromate ion greatly retards this reaction. A hot solution containing 50 g. of platinum and about 5 g. of sodium bromate per liter can be made alkaline to sensitive litmus paper by the addition of sodium bicarbonate, boiled for a few minutes, cooled, and separated from the precipitated impurities before the solution becomes acid.

By using this method of collectively precipitating impurities, subsequently precipitating the platinum once with ammonium chloride and igniting to sponge, platinum of an estimated purity of 99.99 per cent. was prepared from scrap platinum containing 5 to 10 per cent. of impurities, principally platinum metals. From scrap containing 99 per cent. of platinum, metal estimated to be 99.995 per cent. pure was obtained. Under less favorable conditions metal estimated to be 99.95 to 99.97 per cent. pure was obtained from a lot of Colombian grain platinum containing about 84 per cent. of platinum, 3 per cent. of gold, 8 per cent. of iron, 3 per cent. of iridium, palladium and rhodium combined, and 1 per cent. of silver and copper. Most of the gold had been removed first by a preliminary treatment of the ore with cold aqua regia.

The base metals which are likely to be associated with platinum should be precipitated without difficulty under the conditions outlined. Palladium, rhodium and iridium appear to precipitate very nearly completely. Osmium and ruthenium probably precipitate, except for the portion of the ruthenium that is present as nitrosochloride and the osmium and ruthenium that may be oxidized to the tetroxides by the action of the bromate, if the solution at any time becomes acid enough to liberate bromine. Silver is likely to remain in the solution as dissolved silver chloride and gold is divided between the precipitate and the solution.

However, gold may be eliminated by adding sufficient ferrous sulfate to precipitate it as metal before the addition of sodium bicarbonate. The iron from this reagent, of course, is precipitated with the other impurities and may in fact be helpful in collecting small amounts of other precipitated compounds. Some platinum will appear in the precipitate but not enough of it to detract from the value of the method. Work is now in progress on the reaction of the individual platinum metals with hydroxyl ions as well as other phenomena underlying the procedure, but the observations described above seem to be of sufficient interest to those concerned with the purification of platinum to warrant mention in this paper.

The procedure used is as follows: The solution of impure platinum in aqua regia is evaporated once or twice to eliminate nitric acid in the manner previously described. Enough sodium chloride is added to form sodium chloroplatinate and similar salts of the other metals. The solution is evaporated and the residue thoroughly dried to remove as much as possible of the hydrochloric acid.⁷ The residue is dissolved in water and the solution diluted so as to contain about 50 g. of platinum per liter. If gold is thought to be present, ferrous sulfate solution is added in small portions until no further precipitation of gold occurs. The solution is decanted from the residue and heated nearly to boiling, after which sodium bicarbonate is added in small portions to neutralize most of the remaining hydrochloric acid. When the solution is nearly neutral, as may be judged by diminishing effervescence, 10 to 12 g. of sodium bromate for each 100 g. of platinum is added, only a little being added at first to determine whether the solution is nearly enough neutral not to decompose the bromate. If bromine is evolved, more sodium bicarbonate is added until the addition of a little sodium bromate causes no further evolution of bromine. The remaining sodium bromate is then added and the addition of small portions of sodium bicarbonate continued until the solution just turns sensitive litmus paper blue. The solution is now brought rapidly to boiling and again tested with litmus paper. If it is acid a little more sodium bicarbonate is added and the solution is then boiled for three to five minutes. It is finally tested once more and if not alkaline another small portion of sodium bicarbonate is added and the boiling continued for a minute. The solution is then cooled rapidly in running water. After the precipitate settles, the supernatant solution is siphoned off and passed through a filter to collect suspended precipitate.

⁷ The conversion of chloroplatinic acid to sodium chloroplatinate may also be accomplished by adding sodium hydroxide or sodium bicarbonate directly to the solution. However, because of the difficulty of judging when the neutral point is being approached and the consequent danger of adding too much alkali, it is much simpler to add sodium chloride and evaporate to dryness.

If observations or tests indicate that the reaction has not been successful, the solution is acidified with hydrochloric acid, evaporated down and the whole process repeated, after adding a little aluminum chloride. The aluminum salt is added so as to produce a precipitate which will collect smaller amounts of precipitated compounds. This is also done when practically pure platinum is being treated. If the separation of impurities appears to have been successful, the solution is digested with hydrochloric acid to decompose the bromate and the platinum is precipitated with ammonium chloride. The ammonium chloroplatinate precipitated from this solution will carry down considerable sodium chloride, most of which may be removed by leaching with water after the salt has been ignited to sponge.

The precipitate obtained by the treatment with sodium bicarbonate is dissolved in hydrochloric acid and the solution worked over for such metals as it may contain. More or less platinum will be present, depending on the size of the precipitate and other conditions, but the amount should rarely exceed 5 per cent. of the platinum in the solution.

Determining Purity of Platinum

The purity of platinum obtained in a refining process may be tested by one or more methods, singly or in combination. In this laboratory the filtrate obtained from the precipitation of ammonium chloroplatinate is evaporated until the solution is practically saturated with ammonium chloride at room temperature. Most of the remaining platinum separates as a second crop of the salt. The filtrate from this salt is treated with zinc to precipitate residual noble metals. This small precipitate is then examined to determine the identity and approximate amount of the various metals present. While this examination can not yield quantitative results, because some of the impurities in the original material undoubtedly have been retained by the principal crop of ammonium chloroplatinate and more of them by the second crop, it nevertheless offers a useful indication as to the progress of purification. It is certainly safe to estimate that half or more of the total quantity of impurities present in the original material will appear in the zinc precipitate. When no more impurities are found the platinum salt may safely be regarded as pure.

Another very useful method is the thermoelectric comparison of a small sample of purified platinum with a standard sample. This method does not identify the impurities but gives a good indication of their approximate amount. The standard sample, of course, should be the purest platinum available. The test is made as follows: A small portion of the ammonium chloroplatinate is ignited to sponge and the sponge melted on a piece of lime of good quality under an oxyhydrogen flame,

precautions being taken to have a thoroughly oxidizing flame.⁸ The fused button of platinum, weighing three to five or more grams is freed from adhering lime by means of hydrochloric acid and rolled to a thin ribbon. The ribbon may then be cut lengthwise and the pieces joined by fusion, so as to increase the length of the sample. One end of the sample is now fused to the standard sample of pure platinum wire, after which both elements are annealed by momentarily heating them to 1000° or 1200° C. The junction is heated to a known temperature, say 1200° C., and the e. m. f. measured by a sensitive millivoltmeter; or, better, by a potentiometer. In this case it is not necessary to maintain a cold junction at 0° C., since the difference between this temperature and that of the room will not appreciably affect the results. The e. m. f. observed may be interpreted in terms of a particular impurity, such as iridium or rhodium, by reference to a chart on which is plotted an isotherm of e. m. f. against composition.⁹ The metals which are apt to be associated with platinum, with the exception of gold, invariably cause a positive e. m. f. in the alloy. Small amounts of gold cause a negative e. m. f. One must therefore be sure that gold is absent if the results are to have any significance. It is quite useful to employ this method in conjunction with the chemical examination of the mother liquors.

It may be well to mention that when it is inconvenient or undesirable to ignite ammonium chloroplatinate to platinum in the several steps of the process of purification, the salt may be redissolved by suspending it in hot water and treating it with a current of sulfur dioxide. The first reaction is the reduction of the quadrivalent salt to the salt of bivalent platinum, which is much more soluble. The action of sulfur dioxide also results in the formation of sulfite compounds. These may be destroyed by evaporating the solution with strong hydrochloric acid, although some time is required for this process, as the sulfite compounds are rather stable. Ammonium chloroplatinate may then be reprecipitated by the addition of chlorine or of hydrochloric and nitric acids.

⁸ Unless a thoroughly oxidizing flame is used the sample will be contaminated with calcium. The same thing happens if refractories other than lime are used. The alloys of platinum with small amounts of calcium or similar elements display a positive e. m. f. against pure platinum, thus vitiating the results of the test.

⁹ The e. m. f. of an alloy containing 99 per cent. of platinum and 1 per cent. of rhodium, against pure platinum at 1200° C., may be taken as 2.2 millivolts. Similarly, the e. m. f. of an iridium alloy of 1 per cent. against pure platinum at 1200° C. may be taken as 3.2 millivolts. The curves between pure platinum and the 1 per cent. alloys may be plotted as straight lines.

For the thermoelectric properties of alloys containing more than 1 per cent. of either metal, see W. Geibel: Über einige elektrische und mechanische Eigenschaften von Edelmetall Legierungen. *Zisch f. anorg. Chem.* (1911) 70, 248; also A. L. Day and R. B. Sosman: High-temperature Gas Thermometry, Carnegie Inst. Pub. 157 (1911) 121.

Some ammonium chloride is also added to decrease the solubility of the ammonium chloroplatinate. This procedure has one advantage in that gold which may be present is quantitatively reduced to metal and is therefore readily eliminated. Reprecipitating the platinum salt by means of chlorine or otherwise oxidizing the bivalent compound is not so favorable to the removal of iridium, since this element also is rather easily reoxidized to the quadrivalent state. Palladium, rhodium and base metals should be eliminated about as well by this method as the other. Ammonium chloroplatinate may also be dissolved by boiling with oxalic acid or ammonium oxalate, but this reaction is undesirable because of the formation of double oxalates which are relatively insoluble and difficult to break up in any case.

PALLADIUM

The impurities found in commercial palladium depend somewhat on the source of the metal. Palladium recovered in the refining of copper or nickel may contain silver, gold and platinum, as well as copper and nickel, and smaller amounts of tellurium, selenium, antimony, bismuth, lead and iron. Palladium recovered in the refining of crude platinum may contain small amounts of all of the platinum metals, silver, gold, and base metals, such as copper, iron, lead and zinc. The impurities in scrap palladium depend, of course, upon the previous use of the metal. The method which has been most widely used in the purification of palladium is the repeated precipitation of the compound known as dichlorodiammine palladium or palladosamine¹⁰ chloride. This method is rapid and convenient and may be used to prepare palladium of any desired degree of purity. In some cases it may be desirable to combine with it one or more precipitations of the metal as ammonium chloropalladate.

If the palladium is not already in solution, it is dissolved in aqua regia, using the same proportion of acids as described under the purification of platinum. Palladium dissolves much more rapidly than platinum. The reaction should be conducted in a covered porcelain dish which may be heated if necessary. It sometimes happens that palladium sponge contains some palladium oxide. This compound dissolves with the greatest difficulty in aqua regia and may therefore be found in the undissolved residue. The solution is evaporated on the steam bath to expel most of the excess of acid, after which the sirupy residue is dissolved in water and a little hydrochloric acid. The solution of bivalent palladium chloride thus obtained is diluted so as to contain 40 to 50 g. of palladium per liter.

¹⁰ The preparation and properties of this salt are described in detail in Fremy's *Enc. Chimique*, 30, Pt. 3, 94-98.

If much silver was present, most of it will settle out as silver chloride, the remainder being kept in solution. Inasmuch as the process of purification involves a strong chloride solution and an ammoniacal solution alternately, several reprecipitations may be necessary to eliminate all of the silver.

After filtering off the silver chloride and other insoluble material, ammonium hydroxide is added to the solution until no further precipitation of the flesh-colored compound $[\text{Pd}(\text{NH}_3)_4]\text{Cl}_2 \cdot \text{PdCl}_2$ occurs. The solution is then heated on the steam bath for 15 to 20 min. to redissolve the precipitate. It may be necessary to add a little more ammonium hydroxide, but an excess of this reagent is to be avoided because of the heating which results when the solution is subsequently acidified with hydrochloric acid. Iron and certain other impurities will be precipitated as hydroxides or basic salts. The color of the ammoniacal solution furnishes an indication of the amount of copper that may be present. If the solution is pale yellow or straw-colored, very little if any copper is present. If it is blue or bluish-green, the amount of copper may be considerable and the number of reprecipitations necessary to remove all of it will be increased accordingly. If much copper is present, it may possibly be desirable to precipitate the palladium first as ammonium chloropalladate. It is probable that copper is more rapidly eliminated in this way than by the repeated precipitation of dichlorodiammine palladium, but the platinum metals are much more rapidly eliminated by the latter method.

If a great excess of ammonia has been added inadvertently, some of it may be removed by heating on the steam bath for a time. The solution is then cooled and acidified with hydrochloric acid, which causes the conversion of the soluble tetrammine-palladium dichloride $[\text{Pd}(\text{NH}_3)_4]\text{Cl}_2$ into the relatively insoluble dichlorodiammine palladium $[\text{Pd}(\text{NH}_3)_2\text{Cl}_2]$. Hydrochloric acid is added until no more of the yellow salt forms. A large excess of acid and much heating are to be avoided, as the ammine compound may otherwise be partially decomposed with a resulting unsatisfactory yield of precipitate.

The yellow salt settles rapidly and is separated from the mother liquor by draining on a suitable suction filter, such as a Büchner funnel, and washing with cold water. For the next step in the purification, the salt is suspended in water and redissolved by adding ammonium hydroxide and heating. The process of reprecipitation is repeated as often as may be necessary. For the last step, it may be desirable to avoid contamination with the impurities which are apt to be present in ammonium hydroxide by suspending the salt in water and passing in a current of ammonia generated from a flask of ammonium hydroxide.

If palladium of the very highest purity is to be prepared, it is probably desirable to follow the several reprecipitations of dichlorodiammine

palladium by one or more precipitations of ammonium chloropalladate. If this is to be done, the yellow salt is dried and ignited to sponge in an atmosphere of hydrogen. The sponge is dissolved in aqua regia and the excess of acid eliminated by evaporation as before. The residue is diluted and filtered to remove silica or other insoluble matter which may have been introduced by the previous operations involving alkaline solutions. The volume of the solution is adjusted to contain 100 g. or more of palladium per liter. An equal volume of saturated ammonium chloride solution and one-third as much nitric acid (sp. gr. 1.42) are added and the whole solution is then digested in a covered dish on the steam bath, until the palladium is precipitated as the crystalline red salt, ammonium chloropalladate, $(\text{NH}_4)_2\text{PdCl}_6$. The precipitation can be made so nearly complete as to leave a mother liquor which is almost colorless. The salt is filtered off on a Büchner funnel and washed with a 20 per cent. solution of ammonium chloride. If a second precipitation is to be made, the salt is ignited to sponge and redissolved in aqua regia. As was stated previously, this method of purification aids in the removal of base-metal impurities, particularly those which remain dissolved in an ammoniacal solution, but does not tend to eliminate platinum metals rapidly.

As in the case of platinum, the progress of purification can be followed by examining the metals remaining in the mother liquors from each precipitation. To do this the mother liquors are evaporated to dryness on the steam bath and the residues digested with nitric acid to destroy ammonium chloride. After this treatment, the solution is again evaporated to dryness and the residue transferred completely to a crucible and ignited under hydrogen. This is necessary to decompose ammine compounds, of platinum and some of the other metals, which do not respond to the usual qualitative reactions for these metals. After the ignition, the residue is digested with aqua regia and examined for such metals as are thought to be present. Platinum and copper are likely to be the last metals to be eliminated.

Either salt obtained in the process of purification can be ignited directly to palladium sponge. It is desirable to do this in an atmosphere of hydrogen. The use of hydrogen prevents the oxidation of palladium, which would otherwise occur. On the other hand, the metal should not be cooled in hydrogen because the palladium sponge takes up so great an amount of this gas. The current of hydrogen is therefore interrupted after the ignition is complete but before the metal is cooled. In cooling, the sponge will be partly oxidized but this can be corrected by moistening it with formic acid and drying at 100° to 150° C., or by wetting the sponge with alcohol, igniting the latter, and allowing it to burn off spontaneously. A repetition of either treatment may be neces-

sary. Completely reduced palladium sponge has a uniform light gray color.

RHODIUM

The chief source of rhodium is in the residues from the refining of grain platinum. The crude metal is likely to contain any or all of the platinum metals, especially iridium, as well as base metals such as iron, lead, zinc, tin, etc.

Two compounds which have been used for the purification of rhodium are the relatively insoluble chloropentammine rhodium chloride, also known as purpureo rhodium chloride, $[\text{Rh}(\text{NH}_3)_5\text{Cl}]\text{Cl}_2$, and potassium rhodium nitrite, $\text{K}_3\text{Rh}(\text{NO}_2)_6$, also relatively insoluble. It is quite likely that several reprecipitations of the compound first named will result in the elimination of other platinum metals. It was found, however, that the salt is not nearly as suitable for purposes of refining as the ammine compound used in the purification of palladium. It is not readily redissolved in ammonia and the percentage yields of salt are less, so that only a small proportion of the original material is obtained as purified metal if the precipitation is repeated a few times. Furthermore, lead is a persistent impurity. Potassium rhodium nitrite is much more conveniently handled but the precipitate is accompanied by impurities which are thrown out by hydrolysis in the neutral or slightly alkaline solution. This disadvantage was overcome by forming first the much more soluble sodium rhodium nitrite, filtering off impurities precipitated by hydrolysis, and then adding ammonium chloride in order to form the relatively insoluble ammonium rhodium nitrite. The latter compound is readily decomposed by hydrochloric acid, yielding a solution of rhodium chloride. As will be shown later, it was found necessary to combine with this method a treatment with sodium sulfide, chiefly to remove lead. Lead and iridium proved to be the impurities most difficult to separate completely. The method to be described was used to prepare rhodium sponge, in which no impurities could be detected spectroscopically, from original material containing only 75 to 80 per cent. of rhodium. The chief impurities were iridium, platinum, palladium and lead, with smaller amounts of zinc and ruthenium. Commercial rhodium usually has a greater content of rhodium but the impurities are likely to be of the same kind and the same method of purification is applicable. As in the purification of the other platinum metals, the degree of purity desired determines the number of repetitions of the process.

Crude rhodium is best converted to a soluble compound by mixing the metal with about two and one-half times its weight of sodium chloride and heating to about 600°C . in a current of chlorine. The metal and the salt should be mixed intimately, which may be done by grinding the two together in a ball mill. The mixture is placed in boats in an electrically

heated tube furnace, or other suitable type of furnace, and heated for 2 to 4 hr. in a stream of chlorine, the temperature finally being raised to incipient fusion. Minor amounts of certain other platinum-metal compounds may distil out but can be recovered in the cool portion of the furnace or in a receiving flask containing water. The sintered mass is cooled in the atmosphere of chlorine and is then dissolved in water. Usually there will be some undissolved residue, the amount depending on a number of factors such as the nature and amount of impurities present, the temperature, and the thoroughness with which the metal was mixed with sodium chloride. The treatment of the undissolved residue is repeated until all or nearly all of the rhodium is in solution.

The solution is diluted so as to contain about 40 g. of rhodium per liter. It is filtered from any insoluble residue and heated on the steam bath. Sodium nitrite is added until the color changes from deep red to yellow, after which the solution is boiled for about an hour; 500 to 550 g. of sodium nitrite should be added for every 100 g. of rhodium. This results in the conversion of rhodium and other platinum metals to double nitrites, all of which are soluble with the possible exception of a mixed double chloride and nitrite of iridium described by Lang¹¹ as having the formula $\text{Na}_3\text{IrCl}_6 \cdot \text{Na}_3\text{Ir}(\text{NO}_2)_6$, and being very slightly soluble in water. Most of the base metals present are precipitated as hydroxides or basic salts but some are converted to soluble double nitrites. If the solution was too concentrated, some of the rhodium salt may separate out on cooling. In this case more water is added to redissolve the salt. The precipitate is filtered from the solution and reworked for the small amount of precious metals which it may contain.

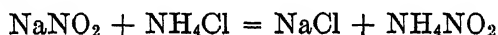
A small amount of sodium sulfide is added to the cold nitrite solution, which is then set aside overnight. This reagent precipitates the lead as well as some of the palladium, platinum, and iridium present in the solution, but very little of the rhodium. Usually 5 to 10 g. of sodium sulfide is enough to treat a solution containing several hundred grams of rhodium. After standing overnight the odor of hydrogen sulfide can usually be detected in the solution. This may be taken as indicating a complete reaction. The sulfides are filtered off and worked up separately for the recovery of the precious metals.

The filtrate is heated to boiling so that any remaining sodium sulfide will react to precipitate rhodium sulfide, which is filtered off. The solution is then treated with an additional 30 to 50 g. of sodium nitrite for each 100 g. of rhodium, so as to make sure that all of the rhodium will be in the form of the double nitrite. The solution is cooled and treated with enough of a saturated solution of ammonium chloride to form the slightly soluble ammonium rhodium nitrite, $(\text{NH}_4)_3\text{Rh}(\text{NO}_2)_6$. This compound is a finely crystalline salt, white when pure, but yellowish if

¹¹ Fremy: *Enc. Chimique*, 30, Pt. 3, 212.

contaminated with other metals of the platinum group. The other metals do not form similar salts which are relatively insoluble, but it is possible that the corresponding iridium salt is isomorphous with the rhodium salt. Some difficulty may be experienced on this account as well as from the possible formation of the mixed double chloride and nitrite of iridium to which reference was previously made. Experience shows that iridium is the most difficult impurity to eliminate after the lead has been separated by the treatment with sodium sulfide. If the original rhodium material contains much iridium, it is well to effect a preliminary separation of this element by precipitation of ammonium chloroiridate, as described in the section of general separations.

In the treatment of the sodium rhodium nitrite solution with ammonium chloride several precautions should be observed. The nitrite solution is usually slightly alkaline and will therefore liberate ammonia when ammonium chloride is added. For this reason a slightly acid solution of ammonium chloride is used. A saturated solution of ammonium chloride containing 10 ml. of hydrochloric acid (sp. gr. 1.18) per liter is satisfactory. The heat of reaction which occurs when the solution of ammonium chloride is added causes some loss of nitrite because of the spontaneous decomposition of ammonium nitrite formed by the following reaction:



This excessive loss of nitrite results in incomplete precipitation of the rhodium. For this reason the solution is cooled in running water, preferably below 18° C., during and after the addition of ammonium chloride. The precipitate of the ammonium rhodium nitrite is allowed to remain in the solution for about an hour, after which it is filtered off on a Büchner funnel and washed with cold water. If the salt remains in the solution for a longer time, it is quite likely to be contaminated because of the slow precipitation of certain impurities, especially iridium. The filtrate is heated to boiling. After the decomposition of residual ammonium nitrite has ceased, the metals left in the solution are precipitated with hydrogen sulfide.

The precipitate of ammonium rhodium nitrite is prepared for the next step in the purification by digesting it with hydrochloric acid on the steam bath. The acid should be diluted with about four volumes of water to avoid too vigorous a reaction. The solution of rhodium chloride resulting from this treatment is evaporated to a sirup to drive off the excess of acid and is then diluted and treated with sodium nitrite as before. The whole procedure may be repeated as many times as necessary to obtain rhodium of the desired degree of purity. Five or six treatments have been found sufficient to produce metal in which only the slightest traces of impurities could be detected spectroscopically.

Sodium nitrite free from heavy-metal impurities is not always available. It is therefore desirable to remove heavy metals from the reagent used for the final steps of purification as follows: Five to 10 g. of sodium sulfide is added to a concentrated solution of 5 kg. of sodium nitrite. Heavy metal sulfides separate out when the solution is allowed to stand overnight. The excess of sodium sulfide is removed by adding a slight excess of a solution of ferric chloride. The reaction results in the precipitation of all the sulfide. As the nitrite solution is slightly alkaline, the excess of iron will be precipitated also. The solution is heated to coagulate the precipitate, which is then filtered off.

As in the case of the other metals, the progress of purification is followed by examining the metals obtained from the mother liquors resulting from the precipitation of ammonium rhodium nitrite, and also by examining the precipitate obtained with sodium sulfide. Two or three treatments with sodium sulfide have been found sufficient to eliminate lead. When lead can no longer be found, this step is omitted from the cycle of operations.

The final precipitate of ammonium rhodium nitrite may be dried and kept as a source of pure rhodium but is not suitable for direct ignition to metal. It is better to convert it to ammonium rhodium chloride. This is done by dissolving the salt in hydrochloric acid and adding about 5 per cent. more ammonium chloride than is required to form $(\text{NH}_4)_3\text{RhCl}_6$. This salt is very soluble in water but can be precipitated by adding to the solution one and one-half times its volume of 95 per cent. alcohol. The salt is filtered off, washed with alcohol, and dried. The residual rhodium can be precipitated from the solution by hydrosulfide.

Ammonium rhodium chloride may be ignited directly to metal, as in the case of the other similar salts. Unless this is done in hydrogen the metal will be partly oxidized, but may readily be reduced by ignition and cooling in hydrogen.

IRIDIUM

Iridium is the most resistant to attack of any of the platinum metals. The best method of attack for crude iridium, or its native alloys with osmium, is to fuse with an alkaline oxidizing mixture such as potassium hydroxide and nitrate, sodium hydroxide and peroxide, or barium peroxide with or without barium nitrate. Treatment with the last-named reagents is not strictly a fusion, as the mass sinters rather than melts, thus permitting greater latitude in the choice of crucibles. Metal crucibles are but slightly attacked and the heating may be done in clay crucibles. If desired, the barium salts can easily be separated from the resulting iridium solution by precipitation as barium sulfate. Potassium salts usually are to be avoided because of the precipitation of potassium

chloroiridate when the iridium is finally converted to chloride. Fusion with three parts by weight of sodium hydroxide and one part of sodium peroxide, preferably in a silver dish, otherwise in nickel or a thick-walled iron vessel, is probably as convenient and satisfactory a method as any.

If necessary, the metal is first reduced to a finely powdered zinc alloy, as described in the section on general separations. The dried material is added to the mixture of sodium hydroxide and peroxide which has been fused to expel moisture. The fusion is kept at 650° to 750° C. for 2 to 3 hr. and stirred if practicable. The molten mass is poured on an iron plate from which it separates cleanly on cooling. Leaching the melt with cold water dissolves some iridium in the form of a sodium salt but leaves most of it as an insoluble oxide or possibly a sodium salt. Nearly all of the osmium and most of the ruthenium present will be in the alkaline solution but much ruthenium remains with the iridium, as do also the other platinum metals in whole or in part, and base metals such as iron, nickel, etc. Much of the ruthenium remaining with the iridium may be removed by treating the residue with sodium hypochlorite solution. The latter is prepared by saturating a cold 20 per cent. solution of sodium hydroxide with chlorine. This solution is mixed with a smaller amount of sodium hydroxide, say one-fifth its volume, in order to prevent the oxidation of ruthenium to the volatile ruthenium tetroxide. Fresh portions of sodium hypochlorite are added until the washings are nearly colorless. Oxygen is generated freely during this treatment of the precipitate.

The residue is digested with hydrochloric acid on the steam bath for several hours and the resulting solution of iridium chloride decanted. A fresh portion of hydrochloric acid is added to insure extracting all of the soluble iridium compounds. The unattacked residue from this treatment is subjected to further alkali fusions or, if rhodium is predominant, it is mixed with sodium chloride and heated in a current of chlorine as described in the section on rhodium.

The repeated precipitation of ammonium chloroiridate is a suitable method for freeing iridium from most of the impurities it may contain, the notable exceptions being platinum and ruthenium. The chief disadvantage of this scheme is the trouble involved in redissolving the iridium salt. This can be done by suspending it in hot water and saturating with sulfur dioxide. The iridium salt is rather easily dissolved because of its reduction to soluble compounds of trivalent iridium. This method of solution has the further advantage of precipitating much of the rhodium, if more than small amounts are present, as a slightly soluble double sulfite. The resulting solution, filtered from any precipitate, is evaporated with an excess of hydrochloric acid to decompose the sulfite compounds. The residue is taken up with water and treated with chlorine or aqua regia, ammonium chloroiridate being precipitated.

Some ammonium chloride is added to the solution to replace any of this salt that has been destroyed and to decrease the solubility of the double salt. Ammonium chloroiridate can also be decomposed by prolonged heating with aqua regia or chlorine. This treatment may be more tedious than the use of sulfur dioxide but it results in the complete destruction of the ammonium radical and also tends to remove small amounts of osmium as the volatile tetroxide.

For obvious reasons, platinum can not be eliminated by repeated precipitation of ammonium chloroiridate. Ruthenium persists with the iridium also, in spite of the greater solubility of its ammonium salt. Most of the ruthenium present with iridium can be removed by pouring the solution of the chlorides into an excess of sodium hydroxide, saturating the resulting mixture with chlorine and distilling off ruthenium tetroxide. The details of this treatment are described under ruthenium. Repeating the distillation several times will produce iridium very nearly free from ruthenium but the strictly complete removal of the latter element is very difficult, if not impossible, by this method.

The results obtained by another and simpler method for the removal of ruthenium have been very satisfactory but require additional confirmation. A solution containing about 37 g. of iridium as chloride and 25 ml. of hydrochloric acid in 700 ml. was saturated with hydrogen sulfide at room temperature. It was set aside for about 60 hr., after which the solution was filtered off and treated further with hydrogen sulfide for several hours, the temperature being kept not far from the boiling point. The solution was set aside overnight and then filtered from the precipitated sulfides. The metals precipitated by the two treatments amounted to somewhat less than 15 per cent. of the total metal in the solution. Upon pouring the residual iridium solution, first reoxidized with chlorine, into an excess of sodium hydroxide solution and distilling with chlorine, no ruthenium was found in the distillate. The separation is based on the relative ease with which ruthenium is precipitated by hydrogen sulfide.

The iridium thus freed of ruthenium was subsequently precipitated as ammonium chloroiridate and this salt ignited to iridium sponge. The sponge was mixed with about 12 times its weight of test lead and the mixture heated in an Acheson graphite crucible at about 1200° C. for 12 hr. The resulting melt was cooled and treated with nitric acid to dissolve the excess of lead. The crystalline residue was digested with dilute aqua regia to dissolve any lead-platinum and lead-rhodium alloys that might be present. Neither platinum nor rhodium was positively detected in this solution. It is quite likely that most, if not all, of these metals was removed in the treatment with hydrogen sulfide. Spectrographic examination of the crystalline iridium gave no evidence of the presence of any of the other platinum metals or lead, but did show some

iron and silicon, presumably introduced from the graphite crucible. It should be stated that the spectrographic examination of iridium is not as satisfactory as it is for some of the other metals. The spectrum of iridium is complex and many of the sensitive lines of the other platinum metals are coincident with iridium lines unless great dispersion is used.

Fusion of iridium sponge with lead should be very useful for the removal of platinum, palladium, and presumably rhodium, as these metals form with lead alloys which are soluble in aqua regia. Ruthenium and iron will combine with the iridium in this treatment. A disadvantage of this reaction is that it leaves the iridium in a crystalline condition, so that it is very difficult to put into solution if it is to be further purified.

Work has recently been resumed on the elimination of ruthenium and platinum from iridium. It is hoped that it will be possible to devise a method for the complete separation of platinum by wet methods, thus obviating the fusion with lead with its attendant danger of introducing impurities from the crucible and its other disadvantages. Use is being made of methods of differential reduction, such as treatment with copper or mercury, which have been proposed by Aoyama¹² and by Karpov,¹³ respectively, for the separation of iridium and platinum for analytical purposes. This laboratory has found that solutions of titanous salts will precipitate rhodium and platinum completely in the presence of iridium, in confirmation of the work of Wada and Nakazono.¹⁴ The experiments were on analytical samples, however, and the extension of such a method to the purification of comparatively large amounts of iridium has its difficulties.

Ammonium chloroiridate can be ignited directly to metallic iridium. If done in air the sponge is partly oxidized. This is easily corrected by igniting and cooling the sponge in hydrogen.

RUTHENIUM

Ruthenium is one of the few metals forming volatile compounds which can be distilled from solution. Because of this property, ruthenium is rather easily separated from base metals and from the metals of the platinum group with the exception of osmium. Ruthenium and osmium are converted to tetroxides when alkaline solutions containing compounds of these metals are saturated with chlorine. Both of the tetroxides can be distilled from such solutions at or near the boiling point. The separa-

¹² S. T. Aoyama: Quantitative Separation of Platinum and Iridium. *Jnl. Chem. Soc. Japan* (1923) **44**, 427; *Chem. Abs.* (1923) **17**, 3146.

¹³ B. J. Karpov: New Method of Separating Platinum and Iridium. *Ann. Inst. Platine* No. 4 (1926) 360; *Chem. Abs.* (1927) **21**, 3168.

¹⁴ I. Wada and T. Nakazono: On a New Method of Separation of Iridium from Rhodium and Platinum and the Separation of Platinum and Rhodium. *Sci. Papers, Inst. Phys. and Chem. Research, Japan* (1923) **1**, 139; *Chem. Abs.* (1926) **20**, 1040.

tion from osmium is facilitated because osmium can be removed as the volatile tetroxide by distillation with nitric acid. Ruthenium tetroxide is not formed under these conditions.

Ruthenium chloride which has been collected in the manner described in the section on general separations may be converted directly into an alkaline ruthenate solution by adding sodium hydroxide. If the metal has been separated as ammonium chlororuthenate the salt should be decomposed with aqua regia or with chlorine. The ammonia, which would be liberated by the addition of an alkaline solution to the salt, is likely to react with the chlorine used in the subsequent treatment, to form nitrogen chloride, which is spontaneously explosive. According to Howe and Mercer,¹⁵ metallic ruthenium can be put into solution directly by treatment with a strong solution of sodium hypochlorite containing some sodium hydroxide. Some difficulty has been experienced in this laboratory with this method of treatment, possibly because the metal was not very finely divided or because certain impurities were present. Ruthenium can be readily converted to soluble ruthenates by fusing the metal with three parts of sodium hydroxide and one part of sodium peroxide at 600° to 700° C. in a suitable container. Silver is preferable for this purpose, but nickel or iron can be used if there is no objection to introducing considerable amounts of these metals. Leaching such a melt with water extracts most of the ruthenium as soluble sodium ruthenate, Na_2RuO_4 . Some of the ruthenium, however, may remain in the residue, especially if other metals, such as iridium, are present. This residue may be dried and fused again in order to complete the solution of ruthenium, or extracted with a solution of sodium hypochlorite as described in the section on iridium.

If osmium is present, which is almost invariably the case, the alkaline solution of ruthenium is first acidified with nitric acid, 5 to 10 per cent. by volume being added in excess, and distilled to effect the removal of practically all of the osmium as tetroxide. For this distillation the solution is contained in a flask of suitable capacity provided with a glass stopper through which pass an inlet tube leading almost to the bottom of the flask, a delivery tube leading to the receiving flasks, and a funnel, provided with a stopcock, for the admission of nitric acid. The portion of the inlet tube immersed in the solution should be as wide as possible to obviate clogging by salts. The receiving flasks are preferably glass-stoppered also, and at least two or three should be used in a chain. After all the nitric acid has been added, the solution is heated and osmium tetroxide distilled as described in the section on general separations.

After the distillation of osmium tetroxide is completed, the solution is cooled, made alkaline with sodium hydroxide and an excess of 20 to 30

¹⁵ J. L. Howe and F. N. Mercer: Solubility of Ruthenium in Hypochlorite Solutions and an Attempt to Utilize the Reactions for the Quantitative Determina-

g. of sodium hydroxide per liter added. The cold solution is saturated with chlorine, after which the temperature is raised to 80 or 90° C. and the current of chlorine continued, but considerably diluted with air. At this temperature ruthenium tetroxide will distil quite rapidly. The receiving flasks contain hydrochloric acid diluted with four volumes of water and one volume of alcohol, except that no alcohol should be added to the first flask. Care should be taken that the delivery tube and the first receiver are not cooler than about 25° to 30° C. to prevent stopping up the delivery tube with crystals of ruthenium tetroxide. When the oily droplets of ruthenium tetroxide no longer appear in the delivery tube, the solution is boiled for a short time, continuing the current of chlorine mixed with air. After the distillation of ruthenium tetroxide seems to have stopped, more sodium hydroxide is added and the current of chlorine continued, whereupon ruthenium tetroxide again distils. This operation is repeated as long as significant amounts of ruthenium tetroxide appear to distil. Small amounts of ruthenium and other metals remaining in the distilling flask may be recovered, as described in the section on general separations. The contents of the receiving flasks contain quadrivalent ruthenium chloride in the solution containing no alcohol and mainly trivalent ruthenium chloride in the solutions containing alcohol. They are combined and evaporated to remove most of the excess of hydrochloric acid, after adding more alcohol if necessary. Some nitric acid is added to the residue and the solution again evaporated to dryness. This should be repeated two or three times to remove any small amounts of osmium. If the original solution has been distilled with nitric acid, the ruthenium solution is not likely to contain enough osmium to warrant its recovery. The residue from the evaporation with nitric acid is evaporated once with hydrochloric acid. The residue is then taken up with water and distilled with chlorine from alkaline solution as before. The second distillation eliminates small amounts of iridium or other impurities which may have been carried over mechanically in the first distillation. The ruthenium solutions in the receiving flasks are again evaporated with nitric acid, to remove any remaining traces of osmium. Finally the residue is evaporated once with hydrochloric acid. It is now principally in the form of ruthenium nitrosochloride, RuNOCl_3 , or the corresponding acid, $\text{H}_2\text{RuNOCl}_5$. This compound forms a relatively insoluble double salt with ammonium chloride similar to the salts of the other platinum metals. The precipitation of the double chloride converts the ruthenium to a form that may be ignited directly to sponge and also eliminates any small amounts of base-metal impurities. If a second precipitation is desired, the salt is decomposed with aqua regia and the resulting solution evaporated to dryness. The residue is taken up in water, filtered from insoluble matter and reprecipitated with ammonium chloride from a small volume. Because of the somewhat greater solubility of the ruthe-

nium salt, it is better to use more concentrated solutions of ruthenium than can be used of the other platinum metals. The salt is filtered on a Büchner funnel and washed with a 20 per cent. solution of ammonium chloride or with alcohol.

Ruthenium sponge oxidizes on heating in the air. To obtain metal free from superficial oxidation, the sponge is ignited and cooled in a current of hydrogen. The oxidation of ruthenium by ignition in air does not result in the formation of appreciable amounts of volatile ruthenium tetroxide except at high temperatures. When the sponge is melted in an oxyhydrogen flame, the characteristic odor of the tetroxide, somewhat similar to the odor of ozone, is easily detected.

The ruthenium left in the filtrates from the precipitation with ammonium chloride can be recovered by evaporating to dryness, destroying the excess of ammonium chloride with nitric acid and finally igniting the entire residue in hydrogen.

The removal of metals other than osmium during the process of purification may be followed by examining the metals left in the solution in the distilling flask. The absence of significant amounts of osmium can be determined by distilling a portion of the ruthenium solution with nitric acid and receiving the distillate in sodium hydroxide solution containing a little alcohol. Even small amounts of osmium produce a violet-colored solution. Spectrographic examination of the purified metal is probably the best method for determining its purity.

OSMIUM

The distillation of osmium tetroxide from a solution strongly acidified with nitric acid affords a simple and direct way of separating osmium from the other platinum metals and in the absence of halogen acids from all other metals as well. In the fusion of crude osmium or osmiridium with sodium hydroxide and sodium peroxide, as described in the section on iridium, the solution obtained by leaching the melt with water contains osmium as sodium osmate, Na_2OsO_4 , and also contains ruthenium and small amounts of the other platinum metals. Certain base metals, such as tin, lead, and zinc, may also be present. The method of conducting the distillation is described in the section on general separations.

Numerous methods have been proposed for the recovery of osmium from the alkaline osmate solution. Probably the best are to electrolyze the solution,¹⁶ causing osmium dioxide to precipitate at the cathode, or to exactly neutralize the solution with sulfuric acid.¹⁷ The latter reaction also causes the precipitation of hydrated osmium dioxide. The solution

¹⁶ H. Moraht and C. Wischin: Beiträge zur Kenntnis des Osmiums. *Ztsch. f. anorg. Chem.* (1893) **3**, 153.

¹⁷ O. Ruff and F. Bornemann: Über das Osmium, seine analytische Bestimmung, seine Oxyde und seine Chloride. *Ztsch. f. anorg. Chem.* (1910) **65**, 429.

is digested on the steam bath for several hours to coagulate the precipitate and to convert it into a form which is not likely to deflagrate when subsequently ignited. The precipitate of osmium dioxide is dried and reduced to metal in a current of hydrogen. Osmium metal thus prepared is practically free of platinum metals but may contain small amounts carried over mechanically from the distilling flask. The metal will also contain alkali compounds and small amounts of impurities such as iron and silica present in the sodium hydroxide solution. It can be further purified by burning it directly to osmium tetroxide in a current of oxygen. In preparing osmium for use in the determination of its atomic weight¹⁸ the metal was heated in boats in a hard glass tube in an electric furnace. Half of the combustion tube was bent downward at an angle of 45° and the end of the tube held in an open Erlenmeyer flask containing enough hydrochloric acid (sp. gr. 1.12) to seal the opening of the tube. The receiving flask was cooled with ice. When the temperature of the furnace reached 220° to 230° C. a vigorous reaction occurred, accompanied by a rapid absorption of oxygen and a progressive glowing of the metal. The product was a black powder of greater bulk than the metal, and was presumably the dioxide. At this stage it was necessary to supply oxygen rapidly to avoid a diminished pressure in the tube. After this reaction had subsided, the temperature was raised gradually, whereupon the material in the boats oxidized quietly to the tetroxide. The latter condensed to a solid in the cool portion of the tube. When all of the metal was converted to the tetroxide the solid cake was transferred to the receiving flask by gently warming the tube until the mass loosened and dropped into the flask. A modification of this method, in which the hydrochloric acid is omitted and the receiving flask is cooled in a freezing mixture, may be used if osmium tetroxide is the desired product.

The contents of the receiving flask were quickly transferred to a flask fitted with a reflux condenser, only ground-glass joints being used. To this flask hydrochloric acid (sp. gr. 1.12) and a little alcohol were added. The flask was warmed gently, and the temperature raised gradually to incipient boiling in about 3 hr. The osmium tetroxide was gradually reduced to quadrivalent osmium chloride, completion of the reaction being shown by the absence of droplets of osmium tetroxide condensing on the walls of the flask, the change of color of the solution to a clear, transparent red, and the absence of the characteristic odor of osmium tetroxide. The end of the reflux condenser was fitted with a trap containing a solution of sodium hydroxide and alcohol to catch the small amount of osmium tetroxide which passed the condenser. Twice the

¹⁸ R. Gilchrist: The Preparation of Pure Osmium and the Atomic Weight of Osmium. Dissertation, The Johns Hopkins University, 1922. Also R. Gilchrist: New Determination of the Atomic Weight of Osmium, briefly described in *Jnl. Wash. Acad. Sci.* (1923) 13, 46.

theoretical amount of hydrochloric acid required to form H_2OsCl_6 was used.

A similar reaction was used for the preparation of bromoosmic acid, H_2OsBr_6 . When hydrobromic acid is used, alcohol is not needed but may be used to hasten the reduction of the tetroxide.

The solution containing chloroosmic acid was evaporated on the steam bath without loss of osmium. It was necessary to keep the solution distinctly acid to prevent the precipitation of a basic compound, presumably the hydrated dioxide. From this solution the precipitation of ammonium chloroosmate by the addition of ammonium chloride was analogous to the precipitation of the similar salts of the other metals. The salt is brick red in color and is relatively insoluble. The salt was ignited to metal in a current of hydrogen, a temperature of 600° to 700° C. being finally attained. The hot sponge was cooled in oxygen-free nitrogen. This metal, preserved in a stoppered bottle, did not for some days begin to emit the odor of osmium tetroxide. As is well known, finely divided osmium unites with oxygen to form the tetroxide even at ordinary temperature. Osmium allowed to cool in hydrogen glows when brought into contact with air, thereby causing some loss of osmium as the tetroxide.

Further purification, if thought necessary, can be accomplished by repeating the cycle of operations just described, beginning with the sponge obtained by the ignition of the ammonium chloroosmate. Osmium remaining in the filtrates from the precipitation by ammonium chloride can be recovered by evaporating the solution to dryness and igniting the entire residue under hydrogen.

In working with osmium, consideration must always be given to the danger of poisoning from osmium tetroxide. Even in minute quantities the vapors are very irritating to the mucous membranes. Prolonged exposure to small concentrations, as well as brief contact with larger amounts, may cause serious and possibly permanent injury, especially to the eyes. The compound fortunately has a penetrating, readily recognizable odor, which serves as a warning.

ACKNOWLEDGMENTS

Acknowledgments are made to Messrs. L. R. Kleinschmidt, Howard A. Jones, and Max Hopper, who assisted in the experimental work on the purification of platinum and rhodium, and also to several other members of the staff of the Bureau of Standards, especially in the laboratories for pyrometry, spectroscopy, and chemical metallurgy, who made the physical tests employed in the determination of the purity of the metals and whose interested cooperation materially helped the progress of the work in other ways.

DISCUSSION

J. E. HARRIS, New York, N. Y.—We have made use of a method of separating the impurities from platinum collectively with marked success. At one time we used sodium bicarbonate and sodium hypobromite in our process and with these reagents ran into two difficulties: (1) the precipitate of impurities was colloidal and hard to filter, and (2) the use of sodium bicarbonate caused a precipitate of basic carbonate of platinum to come out with the impurities, thereby decreasing the yield. We have been able to modify the method so as to avoid the use of both the sodium bicarbonate and the oxidizing agent and with this modification have avoided the difficulties mentioned. I should like to ask Dr. Wichers whether he has met with similar troubles in his use of the bromate method.

E. WICHERS.—We had considerable difficulty at first with solutions that would not filter properly, but since we have followed the procedure just as it is outlined in the paper, we no longer have any trouble from that source. Very seldom, I should say, do we have difficulty in getting precipitates that settle readily and filter well.

The use of sodium hypobromite is interesting, because in effect it gives sodium bromate; in a hot solution, hypobromite goes over to bromate. It was through our use of hypobromite that we came to the conclusion that it was really bromate that did the work and kept the platinum from reacting with the alkali. By adding bromate rather than hypobromite, we avoid the evolution of bromine fumes and all that unpleasant business. Apparently, if you take precautions to get the solution neutral to begin with, add the bromate and just enough bicarbonate to turn litmus, boil for 2 or 3 min. and stop, you almost invariably get precipitates that settle well and offer no difficulty in filtering. That is true, at least, for the solutions we have handled, covering quite a range of impurities. I was acquainted, of course, with the method of your laboratory and reference is made to it in the paper.

H. T. REEVE, New York, N. Y.—The authors state that "small amounts of gold cause a negative e.m.f." How much gold is necessary to give zero e.m.f. against pure platinum; also, does silver behave in the same way as gold? The other point I have in mind is the handling of small samples in such a way as to avoid any chance of getting calcium into platinum through melting on lime. We press up the sponge in the form of a cylinder of $\frac{1}{2}$ -in. dia. and stand this on a pressed disk of the same material. One end of the cylinder is melted with an oxyhydrogen torch, then it is turned over and the other side is finished. With a little care, this operation can be performed without welding the sample to the disk, and one can then be sure that the melting operation has not introduced any except gaseous impurities.

F. E. CARTER, Newark, N. J.—Certainly Mr. Wichers and his associates are to be congratulated in covering a tremendous amount of work in a reasonably short paper, for a great deal of information is given. One is struck particularly by the need of attacking differently each step in a purification, depending on what has been found up to that step. It is rather disconcerting to study the large number of operations necessary and the skill that is obviously put into them and then read on the opening page: "It is hoped that this paper will show that any of the metals can be prepared in any desired degree of purity by relatively simple straightforward chemical methods." In my paper,¹⁰ I say, "The preparation of the pure platinum metals from the commercial grades constitutes one of the most difficult problems of inorganic chemistry." There must be some reason, in my belief, for the divergence in those statements. I think refiners are justified in saying that it is a difficult

¹⁰ F. E. Carter: The Platinum Metals and Their Alloys. *Proc. Inst. Met. Div.* (1928) 761.

problem. Possibly it is due to the fact that on the small scale that Dr. Wichers and his associates have been working, they do not encounter the troubles that we have when dealing with many kilograms at one time. That is the only point I would like to make here, and Dr. Wichers has covered that, for he simply said that experience has shown that these methods work; whether they work for the larger quantities must be left open.

I would like to ask Dr. Wichers what he means by reasonable time for purification. The authors say that it brings the time required for purification of any of the metals within reasonable limits. Supposing one had 100 oz. to refine, what would be considered a reasonable time for getting one of the metals in the really pure state?

On page 615, in getting the purity of the platinum wire, testing against standard, the authors say that both elements are completely annealed by momentarily heating them to 1000° or 1200° C. There is a difference in electromotive force whether heat is applied momentarily or whether it is prolonged. The Bureau of Standards, I believe, in making tests of thermocouples that are sent for standardization, anneal electrically for 5 or 6 hr. at a temperature of about 1500° , and we find that this makes quite a difference in the electromotive force. Do you test the purity of palladium, say, against a standard palladium as you do the platinum? I do not know that I have ever seen that stated.

Quoting from page 622: "Ammonium rhodium chloride may be ignited directly to metal, as in the case of the other similar salts. Unless this is done in hydrogen, the metal will be partly oxidized." I do not know whether rhodium oxidizes under those conditions or not. Will you enlarge on that somewhat?

E. WICHERS.—The simple methods and the reasonable time—that is, of course, a little hard to define. Our purpose in making the statement was to encourage those to whom work on platinum metals might seem entirely shrouded in mystery. There is a good deal of that attitude on the part of chemists, due largely to absence of much literature on the subject. After all, while the methods may take a little time, they involve no trick apparatus, no unusual reagents and no very unusual reactions; they are simply recrystallizations of compounds.

As to the length of time, it depends, of course, on the metal and also on one's experience and the degree of purity desired. I believe that purification of platinum from commercial platinum, say 99 per cent., to thermoelement platinum by the use of this collective precipitation of impurities, followed by one precipitation of ammonium chloroplatinate can be accomplished within three days, three shifts of 7 or 8 hr. Whether that seems long or not depends a good deal, of course, upon the point of view. That is an estimate, more or less a guess, based on our own experience. Doing this with 100 oz., of course, depends on the use of suitable equipment.

As to complete annealing, that point is a little misleading. What is meant is that the sample is annealed sufficiently for the purposes of the thermoelectric test. As to the effect of heating for 5 or 6 hr., I am not sure all of that is annealing. I believe there are possibly internal changes of composition occurring in the platinum, depending on the gases or small traces of impurities from the refractories present. I would also say that momentary or flash heating is probably not total annealing.

As to the thermoelectric test for palladium, we have used it and it follows very much along the line of platinum. With palladium as with platinum, you start with an arbitrary standard; there are no fixed values, simply relative values, against some standard which you may select. There are some difficulties with palladium which are not met with platinum but for purposes of refining, control by this test is equally applicable to palladium.

As to the oxidation of rhodium sponge in the air, we have found that that does take place. A case in point, although not bearing exactly on this, is that if rhodium

sulfide is ignited in air it is converted to oxide, the composition being close to Rh_2O_3 . If the oxide did not form in heating the sponge, oxide formed by ignition of the sulfide would also dissociate, would it not, Dr. Carter?

In reply to Mr. Reeve's question about the platinum-gold alloy: We have not determined what percentage of gold in platinum yields an alloy that is thermoelectrically neutral to platinum. An alloy containing 0.25 per cent. of gold gave a negative electromotive force of 135 microvolts to our purest platinum at 1200°C . We do not know what the effect is of small quantities of silver in pure platinum.

T. A. WRIGHT, New York, N. Y.—This paper interests me in that it deals with analysis, purification and recovery. Those three headings coincide in many places and are quite different in others. There is one point I wish to confirm and that is the idea of getting real purity on the ammonium chloride precipitation of the platinum. We have found, as far as palladium is concerned, that it took seven precipitations and reprecipitations before we could get no reaction for palladium by an analytical method. I should like to know if others have had that trouble.

Furthermore, we have had reason to believe that with the gold or the lead alloy method that is mentioned, and which is treated extensively in a paper issued by the Bureau of Standards,²⁰ and also in a Japanese paper²¹—i.e., the separation by the lead alloy method—the solubility of rhodium depends a good deal on the ratio of the gold, rhodium and lead. I would like to get some corroboration on that.

In regard to the silver chloride mentioned, there is a tendency for the silver chloride to carry down palladium and that brings up another interesting point on the lead button. It has been found that when you dissolve the lead button in nitric acid it is desirable to get rid of the bulk of the lead with sulfuric acid and that this mass or precipitate of lead sulfate does not seem to carry any appreciable amounts of precious metals, but when we afterwards may fume that solution down to remove the remainder of the lead, the lead sulfate residue does carry precious metal with it. I think that Dr. Wichers has looked at this more from the standpoint of purification and possibly the quantitative feature was not so important in those operations as was quality. In the analyses you must try for both. I do not know which we worry most about, getting too high or too low results. Lead is a bugbear with us when the lead method is used. It seems to carry through especially into the platinum.

I noticed that the authors used nitric acid in the osmium recovery. We have used chlorine instead and have a battery of flasks, all with ground-glass joints. We had trouble for a while with leakage, but we stopped the leakage, and did not get any organic compound to give us trouble with the chlorine, by just borrowing a little talcum powder from one of the stenographers. It made a very tight joint on the ground-glass tube.

Relative to ruthenium, I judge the authors did not get into much trouble with volatile compounds. We have always worried more or less about that in washing with hypochlorite solution. I think seven years ago we left some of that overnight with a filter paper on top of it and it was black the next morning, or almost black, acting very similar to osmium in that respect.

E. WICHERS.—In our opinion, the methods suggested here for purification are largely not applicable to analysis. There are two distinct objectives and the same methods are not suitable for both. As to the effect of sodium hypochlorite on ruthenium, volatilization of ruthenium tetroxide will occur if the alkaline solution is saturated with chlorine. If you have a little excess sodium hydroxide present, the

²⁰ R. Gilchrist: Determination of Iridium in Platinum Alloys by the Method of Fusion with Lead. U. S. Bur. of Standards *Spec. Pub.* 483 (1924).

²¹ S. Aoyama: *Op. cit.*

ruthenium is oxidized to perruthenate but not to the tetroxide. There is no loss. That was pointed out by Professor Howe and his associates²² in a paper about two years ago.

E. L. JORGENSEN, Irvington, N. J.—Even though the same methods may not be generally applicable to both analysis and purification of the platinum metals, I was glad to hear the analytical side of the question introduced. I believe that the idea does exist that the methods used for the two purposes are similar. There is great reluctance in publishing refining methods and analytical methods are classified with them. There also may be the unfortunate tendency on the part of individuals to retain experiences as personal knowledge. The analytical methods used today are very far from giving commercially satisfactory agreements between different assayers. Lack of publicity is greatly to blame for this. I welcome this paper and this discussion as an entering wedge for more publicity of idling knowledge to the benefit of all.

T. A. WRIGHT.—Mr. Wichers, has it been your experience, when evaporating chloride solutions for the removal of nitric acid, and gold-platinum is present, that often on such evaporation free gold separates out? We have found it so on a number of occasions, but never when there was considerable copper in solution.

E. WICHERS.—We have no observation on it. Certainly, gold is very sensitive to reduction by organic matter or even by excessive heating, as by baking on the steam bath.

T. A. WRIGHT.—I had never seen that mentioned in the literature at all. If the platinum is to be precipitated in the presence of the gold, as is sometimes done, this tendency for the gold to separate is of importance. I do not know that copper in such amounts as we have found to inhibit the precipitation of the gold is occluded very much by the ammonium chloride precipitate, although that is only used for a finer precipitation.

²² J. L. Howe and F. N. Mercer: *Op. cit.*

Refining and Melting Some Platinum Metals

By J. O. WHITELEY* AND C. DIETZ,* YORK, PA.

(New York Meeting, February, 1928)

IT IS difficult to give a refining outline that may be followed for any and all combinations of the platinum metals; different combinations require different methods of attack. This paper does not pretend to go into all of the reactions of the metals of the platinum group, but will outline the commercial refining of platinum scrap.

TYPICAL PLATINUM SCRAP

As a concrete case, let us take 100 oz. of miscellaneous scrap, made up of jewelry scrap, dental scrap and laboratory ware. This lot will contain some platinum, palladium, iridium, ruthenium, rhodium, gold, silver and copper.

The first problem is to get all of the metals in solution. If the scrap is principally platinum, palladium and gold, a treatment with aqua regia is the course to pursue; but if the scrap contains alloys high in iridium or ruthenium, aqua regia will act too slowly. In the case in hand, we are going to presume that we have scrap high in iridium and ruthenium, and proceed accordingly.

OBTAINING RESIDUE A

Place the scrap, together with six times its weight of granulated test lead, in a solid carbon crucible. This crucible may be placed inside a clay graphite crucible packed with charcoal, to prevent the excessive oxidation of the carbon crucible by the furnace blast. Place a tight cover on the carbon crucible; heat gradually to a good red heat and allow it to soak in the heat for 4 or 5 hr. After the leaded mass is cool, dissolve in dilute nitric acid (1-6) on a steam plate or sand bath. This dissolves practically all of the lead and most of the palladium, silver and copper, together with some of the platinum and rhodium, and leaves the iridium, ruthenium, gold, and most of the rhodium and platinum as a finely divided residue. Siphon off the clear solution and treat the residue with another portion of dilute nitric acid to insure the solution of the lead and copper. The second nitric acid solution is added to the first, and the residue is filtered on suction and washed thoroughly with hot

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water. This is Residue A, and contains gold, iridium, ruthenium, platinum, rhodium, and some palladium.

It is very important to get the iridium, ruthenium and rhodium into a finely divided condition in order to effect their dissolution and purification by the method which will be outlined later. The lead fusion and nitric acid parting serve to accomplish this division.

TREATMENT LEADING TO RESIDUE B

Add sulfuric acid to the filtrate from the lead parting to precipitate the lead; filter and wash with cold dilute sulfuric, discarding lead sulfate. Treat filtrate from the lead sulfate precipitate with hydrochloric acid, to precipitate the bulk of the silver. Filter off silver chloride. The filtrate will contain palladium and copper, and small amounts of platinum and rhodium which may have gone into solution in the dilute nitric acid treatment. To avoid the tedious evaporation often resorted to in reclaiming these metals from a bulky filtrate, add ammonium chloride salt in generous amount and saturate the solution with chlorine gas. This will precipitate all of the platinum and palladium as double ammonium chlorides, which can be filtered off and washed with cold ammonium chloride solution, and finally with alcohol, dried and burned to sponge. This sponge is added to Residue A.

The filtrate, which contains rhodium and copper, is cemented for several days with copper sheets. This cements out the rhodium, which is filtered off and set aside for further treatment. Call this rhodium black.

An optional method for handling the nitric acid solution from the lead parting is to cement immediately on copper sheets. This cements out silver, palladium, platinum and rhodium, and leaves copper and lead in solution. These cements are filtered off and added to residue A. The advantage of this optional method is to avoid the filtering and washing of a very bulky lead sulfate precipitate. On the other hand, this cementation must be continued for several days to be complete.

Residue A is dissolved in a mixture of three parts hydrochloric, one part nitric, one part sulfuric and two parts water. (The reason for the inclusion of sulfuric acid in this mixture will appear later.) The dissolution is carried out in large cylindrical crocks made of Pyrex glass or of porcelain. The crocks should have close-fitting covers, to retard the loss of nitrous oxide and chlorine. Treatment is best carried out on a steam plate; heating to boiling unnecessarily expels the active chlorine. After action has ceased, allow the solution to settle and siphon the clear solution into large porcelain basins. If sufficient acid has been used, a second acid treatment of the residue is unnecessary. The residue is filtered off and washed with hot water. It contains almost all of the

iridium and ruthenium, and some of the rhodium, but practically no platinum, palladium and gold. This residue is dried on a steam plate, and brushed from the filter paper. The paper is incinerated at a low heat, and added to the brushed off residue. This is Residue B.

EXTRACTING THE GOLD

The aqua regia solution, obtained above, contains platinum, palladium, gold, rhodium, and small amounts of iridium and ruthenium. This solution is evaporated on sand baths to expel nitric acid. The presence of sulfuric acid in the solution serves to expel nitric acid in one evaporation and keeps the solution from evaporating to a dry salt, which might cake hard and crack the basins. It converts any lead that may be present to lead sulfate. After this solution has evaporated to a sirupy consistency, it is cooled and the salt is dissolved in a generous quantity of cold dilute hydrochloric acid (1-10). The solutions are then heated to boiling to dissolve the last traces of salt. The solutions are transferred to a large crock and allowed to settle until clear; settling overnight is good practice. The clear solution is siphoned into another crock, the residue is filtered off and the filtrate added to the clear solution in the crock. The residue will contain silver chloride, lead sulfate and some baked-out gold.

The baked-out gold may be quickly extracted by treating the residue with concentrated hydrochloric acid and careful additions of dilute sodium chlorate solution. Stir vigorously while adding the sodium chlorate solution, in order to avoid explosions. After the gold has dissolved, the solution should be filtered and the filtrate added to the main solution in the crock. The main solution is now saturated with sulfur dioxide gas, which precipitates all the gold as finely divided metallic gold. The gold is filtered off and leached thoroughly with hot hydrochloric acid, to dissolve occluded platinum metals. This leach is added to the filtrate. The gold is then washed successively with hot water, aqua ammonia, and again hot water. These washings are discarded; the gold is dried and melted.

OBTAINING PLATINUM SPONGE

The filtrate from the gold precipitation contains platinum, palladium, rhodium, and some iridium. It is thoroughly saturated for several hours with chlorine gas to counteract the reducing action of the sulfur dioxide treatment and to oxidize the platinum metals in the solution to their highest valency. The solution is now put into basins and evaporated to smaller volume, or until there is no perceptible odor of chlorine gas but there is a strong odor of hydrochloric acid. A small quantity of alcohol is stirred into the hot solution; this serves to reduce the chlorides

of iridium and palladium to a lower valency and to hold them in solution while the platinum is being precipitated. Precipitate the platinum from a hot solution with hot concentrated ammonium chloride solution. Using both solutions hot will give an immediate crystalline precipitation of orange-colored platinum ammonium chloride, which is easy to filter and wash. (The color of the platinum ammonium chloride salt varies from light yellow to orange according to the heat of the reacting solutions.) Filter off the platinum ammonium chloride on suction filters, wash with half and half cold water and ammonium chloride solution, and finally wash with alcohol, dry and burn. A slow, careful burning of the platinum salt is necessary, to prevent loss of platinum as the ammonium chloride sublimates off. Burning under a cover, with the introduction of hydrogen, will aid in avoiding the loss of platinum. This gives a quite pure platinum sponge, but if higher purity is desired, a second crystallization of the platinum may be made.

We might say, at this point, that all of these ammonium chloride precipitations of the platinum metals are somewhat fractional. The precipitated metal always contains small amounts of the other platinum metals, and requires a second crystallization if high purity is desired; and secondly, all of the filtrates contain traces of the metal being precipitated, and must be worked for recoveries.

RESIDUE C

The filtrate from the platinum ammonium chloride precipitation is reoxidized by saturating with chlorine gas, and a little more ammonium chloride salt is added. This precipitates all of the palladium and iridium, and any platinum which did not previously precipitate, as a red double ammonium chloride salt, and leaves whatever rhodium is present in the filtrate. This filtrate is cemented on zinc and the cement is added to Residue B. The precipitated salt is filtered on suction, and washed with a saturated solution of ammonium chloride. It is then washed into a suitable vessel, using a wash bottle and water, and is treated with an excess of concentrated aqua ammonia; no heat is used in this treatment. The aqua ammonia dissolves palladic ammonium chloride to a colorless solution. The undissolved residue contains any platinum and iridium which may be present, together with traces of occluded palladium and base-metal hydroxides. This residue is filtered off and burned, and is called Residue C.

PRECIPITATING PALLADIUM

The ammoniacal palladium solution is diluted by twice its volume of cold water; hydrochloric acid is added to precipitate the yellow salt of palladosamine chloride. This is filtered on suction, washed with cold

dilute hydrochloric acid, and finally with alcohol, dried and burned under hydrogen to palladium sponge. The filtrate from the palladium precipitate is again chlorinated, any salt separating out is filtered off and the filtrate finally cemented with zinc. This cement is added to Residue C and reworked.

A suggested method for reworking cement together with Residue C, which should carry only small amounts of platinum, palladium and iridium, is as follows: Burn, dissolve in dilute aqua regia, filter off the residue, which is iridium. Add this residue to Residue B. The filtrate is evaporated to dryness, taken up in dilute nitric acid and all the palladium present precipitated in hot solution with mercuric cyanide. The white palladium cyanide is filtered off, washed with hot dilute nitric acid, and burned to palladium sponge. The filtrate from the palladium cyanide precipitate is evaporated three times with hydrochloric acid, and the platinum precipitated with ammonium chloride solution, filtered off and burned to platinum sponge. The filtrate from this platinum precipitation, which may contain iridium, and possibly rhodium, is treated, while hot, with hydrogen sulfide gas; the sulfide precipitation is filtered off, burned and added to Residue B.

REWORKING RESIDUE B

Residue B and rhodium black are combined; this combination contains the iridium, ruthenium and rhodium, and traces of other metals. It is intimately mixed with its own weight of sodium nitrate and 10 times its weight of sodium peroxide. This mixture is fused in an iron pot; the fusion should be carried on quietly for several hours, stirred with an iron rod, and finally poured on to an iron plate. The cooled mass is treated with water and the solution, together with the suspended oxides, is transferred to a glass retort of convenient size; chlorine gas is bubbled through the cold solution in the retort for about one-half hour, then, while the chlorine is continued, heat is gradually applied to the retort. The ruthenium will distill off as a volatile oxide, and is collected in a series of three Wolff bottles containing concentrated hydrochloric acid. Care must be taken that the solution in the retort remains alkaline; otherwise the ruthenium will be precipitated in the retort from an acid hypochlorite solution, and therefore will not entirely distill off as it should. The ruthenium chloride solutions from the Wolff bottles are united and evaporated to small volume; ammonium chloride salt is added and chlorine gas run in until ruthenium ammonium chloride salt is precipitated. This salt is filtered off and washed with alcohol; dried and burned, under hydrogen, to prevent formation of volatile ruthenium oxides. (Freshly precipitated ruthenium sponge is soluble in sodium hypochlorite solution, if the sodium hypochlorite solution is kept alkaline

by the addition of sodium hydroxide.) The solution and residue remaining in the retort are decanted into a basin and boiled with a slight addition of alcohol. This precipitates the iridium and rhodium, and leaves the platinum in the filtrate. The platinum may be recovered by making the filtrate acid with hydrochloric, and cementing on zinc and reworking in the usual manner.

The iridium and rhodium are filtered off and washed with dilute hypochlorite solution; the filter paper and contents are transferred to a beaker and boiled with concentrated hydrochloric acid. This dissolves the iridium and rhodium. The solution is diluted and the filter paper residue is filtered off and discarded. There should be no metallic residue. If there is, another caustic fusion is indicated. Concentrate the iridium-rhodium solution to small volume, add ammonium chloride salt, saturate with chlorine gas and filter off the iridium ammonium chloride which is precipitated. Wash with a saturated solution of ammonium chloride. To free the iridium salt from traces of occluded rhodium, dissolve the salt in hot water, concentrate and reprecipitate iridium with ammonium chloride and chlorine gas, filter and wash with ammonium chloride, and finally with alcohol. This salt should be dried carefully and burned to iridium sponge, and finally burned at a high temperature and reduced under hydrogen.

The filtrate contains rhodium. It is cemented on zinc or copper; the cement is dried, not burned, and the paper only incinerated. The rhodium cement is fused with sodium acid sulfate. This fusion dissolves the rhodium and leaves any platinum and iridium undissolved. The fusion is taken up in very dilute sulfuric acid, and rhodium is recemented on zinc.

SUGGESTIONS FOR REFINING

1. As previously stated, the order of procedure may be varied according to the composition of the alloy in hand. For instance, with an alloy high in palladium, we recommend the separation of palladium previous to the separation of platinum.

2. In all cases where gold is present, we recommend the removal of gold first, to avoid its inclusion in the precipitations of the platinum metals. The precipitation of gold with sulfur dioxide gas is quantitative, and yields gold of very high purity. The presence of free hydrochloric and sulfuric acids in the solution during the gold precipitation serves to prevent precipitation of platinum and base metals with the gold.

3. Solutions from which precious metals have been extracted should not be thrown away without a very careful test for platinum metals, as solutions that are in a reduced condition, especially amine solutions, do not react to the usual tests for the presence of platinum metals. Oxidation with chlorine gas, followed by cementation on zinc, may be resorted

to; or saturation with hydrogen sulfide gas; or, best of all, a sample of the solution may be evaporated to dryness, the evaporated salt carefully burned and the residue extracted with aqua regia and tested.

4. Chlorine gas will prove a valuable reagent in platinum refining. Its rapid oxidation of metals in solution, and its use in conjunction with ammonium chloride to obtain complete precipitation of the platinum metals, are of prime importance.

MELTING AND WORKING PLATINUM METALS

Platinum metals are commercially melted by means of the oxyhydrogen torch. The size of the torch should be suited to the amount of the metal to be melted. If much melting is to be done, the torch should be fitted with a platinum-iridium tip to prevent base metals from dropping into the melt. Either oxygen and hydrogen, or oxygen and city gas are suitable fuels for this purpose. If city gas is used for large melts, it is advisable to put the gas under pressure by means of a gas booster. Some melting is being done in the electric induction furnace, but the bulk of the melting has been with the torch.

Crucible

The crucible and cover are made of dry, high-burned lime. A good method of preparing the lime block is to granulate dry lime and press it into a cylindrical block, by means of a hydraulic press, using a pressure of about 150 tons to the square inch. After removal from the press, the lime block is fitted into a light sheet-iron casing, which offers a means of holding the block and of fastening it to the pouring stand. The block is then cut out to form a wide, shallow crucible. A similar lime block is pressed for the cover. This is cut out to form an arched roof, and has a hole in the middle of the top for the introduction of the melting torch, and suitable rim vents for the passage of spent gases.

Up to 400 oz. of platinum may be melted at one time in a lime crucible cut from a lime block 9 in. in diameter and 5 in. high, outside dimensions.

For the purpose of pressing this block, a steel plunger and cylindrical steel mold with walls 1 in. thick are used. The cylinder is split vertically into halves, and made slightly elliptical at the parting lines, to facilitate its removal from the lime block.

The lime block should be used immediately, or it will absorb moisture and crack upon the application of heat.

Crucibles made from zirconium oxide are successfully used for platinum melts, but are not as beneficial to the platinum as lime crucibles. Melting in lime has a purifying effect on platinum, for the lime acts as a cupel and tends to absorb small base-metal impurities. In cleaning

up after a melt, the lime may be dissolved away in hydrochloric acid. On the other hand, a zirconia crucible does not absorb moisture as lime does, and may be used repeatedly from day to day, but it does not have the cupelling action of lime and is more difficult to dissolve away.

It is not advisable to melt platinum in ordinary clay, graphite or silica crucibles, because the melted platinum is likely to take up carbon or silicon from the crucible wall and to become brittle.

Melting Procedure

Before the introduction of the metal, the lime crucible should be thoroughly preheated with the torch flame to drive the occluded gases out of the lime. If this preheating is not done, and platinum is melted in the raw crucible, the molten metal will spit considerably in the crucible, on account of gases released from the lime which bubble up through the metal. For this reason, the second melting in a lime crucible is generally quieter than the first.

After the crucible is properly preheated, the cover is raised and the metal introduced. The torch flame is applied through the cover and the flame regulated to carry a slight excess of oxygen. An oxidizing flame is necessary, to keep the platinum from absorbing calcium and also to give a solid casting, free from gas porosity.

The melt may be stirred with a solid carbon rod, for, due to the oxidizing flame, the melt does not take up any carbon. Chemically pure platinum may be melted and stirred in this fashion.

The metal should be quiet before it is poured. A little experience in melting platinum will indicate the proper pouring condition. A well poured ingot will show a down pipe at the top. If there are blowholes in the ingot, a remelting is indicated; in fact, one or more remeltings can do no harm.

Ingot Molds

Melted platinum may be poured into ingot molds made of hard carbon (Acheson graphite slabs, which are machined in halves and clamped together). For wire bars, an ingot 1 in. square by 12 in. long is a convenient size. For plate bars, a flatter ingot is poured. If the ingot mold is given a slight taper, an ingot may be removed by simply inverting the mold.

In this fashion may be melted pure platinum, pure palladium, and alloys of platinum and iridium; platinum and rhodium; platinum, palladium and gold; palladium and gold, etc. On the other hand, platinum alloys high in ruthenium or osmium might better be melted in an electric induction furnace, due to the tendency of ruthenium and osmium to volatilize in an oxidizing atmosphere, and also due to the tendency of

these alloys to absorb gases from the torch flame, which are difficult to expel and which make it difficult to get a sound casting.

Working

The wire bar ingot is heated to a white heat (1200° to 1300° C.) and hot forged under a power hammer. This hot forging tends to compact the metal, close up minute porousness, and to work out the pipe. After forging, the bar is cold rolled and swaged, or drawn to finished size. For drawing through steel dies, platinum may be lubricated by passing it through a box filled with dry powdered soap. For drawing through diamond dies, castor oil or other suitable wet lubricants may be used.

Flat ingots intended for plate are generally given a light cut in a planer, to remove surface imperfections, and then cold rolled to plate or sheet. A preliminary hot forging of plate ingots can do no harm.

If blisters show up in the plate stock, on annealing, a remelt is indicated.

Annealing

Platinum and platinum alloys should be annealed at suitable intervals during the working. The annealing may be done in a muffle furnace with an oxidizing atmosphere; that is, a free access of air. The metal should not be allowed to come in contact with the open flame, particularly if it is a reducing flame, for some of the alloys have a tendency to pick up contamination from the flame, which causes crystalline fractures.

DISCUSSION

E. WICHERS, Washington, D. C.—Reference is made to the sharpness of separation of iridium and platinum in the lead fusion. Ordinarily that is very sharp indeed if the lead is pure, but we have found that the presence of small amounts of zinc in the whole mixture will cause appreciable amounts of iridium to go into solution.

G. F. KUNZ, New York, N. Y.—Do you not have great difficulty with the lead anyway, and also the copper? If the copper is electrolytic, the alloys, I think, are much better than if commercial copper is alloyed. But lead seems to be a very difficult thing to handle and it does carry traces of different metals.

E. WICHERS.—In this case I do not think it is so much a question of zinc in the lead as that zinc may be in the platinum sponge. Frequently the metals are thrown out of solution by means of zinc. If this sponge is then separated by a lead fusion enough zinc is retained to cause serious trouble.

Sampling and Evaluating Secondary Non-ferrous Metals

By T. A. WRIGHT,* NEW YORK, N. Y.

(New York Meeting, February, 1928)

THE SAMPLING of waste materials containing copper, lead and tin has taken on a new significance within recent years, and is of increasing importance, on account of the entry of some of the copper refineries into this field and the tendency towards consolidations among the smelters. In the past, such material was sold on the basis either of a rough or grab sample or of casual inspection of the lot.

The two principal sources of this waste are the processes of fabrication and the scrapping of automobile, electrical, plumbing, railroad and other equipment. In both cases much material can be sorted, salvaged and sold by classification and grade to be melted "as is" or with the addition of new or virgin metal. The increasing care in classifying exercised in the large manufacturing plants tends to increase the proportion of this usable scrap; on the contrary, the increasing use of more complex and special non-ferrous alloys tends to make sorting and grading more difficult in turnings, borings, clippings, etc., and consequently is making sampling of more moment.

In tin, lead, aluminum, and zinc furnace by-products such as ashes, drosses, skimmings, etc., the trade practice of evaluation has not varied much. For instance, tin drosses, battery lead, and aluminum skimmings are ordinarily sold on what is called "metallic button" or "metallic recovery" or "metallic yield." Copper and lead waste, however, is being sold more and more on actual copper and lead contents based on sample and chemical analyses.

This paper will be devoted chiefly to the sampling of materials suitable only for refining and recovery of one or more of the metals present. The methods described are often based on the fact that but one metal, such as copper, is to be paid for. The loss of volatile constituents in melting does not appreciably affect the final copper result, as the "after-melting" assay can be readily calculated to the "before-melting" basis, but if smelter efficiency continues to increase, so that in time zinc, for instance, may be recovered on a paying basis, the method of sampling dirty brass, for example, will have to be changed also.

As a matter of fact, we may expect that improved refining processes will make possible the partial recovery of lead and tin which are now lost.

* Secretary, Lucius Pitkin, Inc.

GENERAL PRINCIPLES OF SAMPLING SCRAP

Mathematics enters very little into the sampling of metallic scrap as such. The drawing of the sample is all important, and no substitute for personal selection made jointly by the representatives of the buyer and seller has yet been found. On the other hand, on copper-bearing material, as it is known in the trade, a definite ratio of the lot may be set aside and true coning, mixing and quartering practiced as in ore sampling. Table 1 shows a rather rough group of types in which selection has the larger importance and Table 2 those of which definite proportions should be taken. A combination method is advisable when judgment is a large factor and conditions admit: for example, in shipments of skimmings, there may be large chunks of metal; if the sample is being drawn by the tenth or fifth shovel or by an automatic mill the large metallics should be set aside, broken up with a sledge, mixed, quartered down and weighed. The final sample for melting should be made up of the proper proportions of fines and the broken metallics.

TABLE 1.—*Rough Grouping of Metallic Scrap When Selection is of Greatest Importance (Class A)*

Scrap Metal	Copper				Free Iron	
	*Sample	Per Cent.		May Contain	Per Cent.	
		High	Low		High	Low
Armatures.....	M-T	25	10	Sn	80	5
Arc lamps.....	M	66	27		50	0
Brass scrap.....	M	85	10		80	10
Breakage.....	M	65	30		35	0
Briquetted metal.....	M	99	45			
Bronze scrap.....	M	90	75	Sn		
Carburetors.....	M	83	73			
Coils—electric.....	M-T	90	17	Sn	66	0
Composition scrap.....	M	85	60			
Copper scrap.....	M	99	87			
Fields.....	M-T	50	25		70	20
Bottoms, furnace.....	M-D	93				
Generators.....	M-T	20	10		80	5
Gaskets.....	M	71	48		15	0
Radiators.....		Flat Rate		Sn-Pb	20	
Tuyeres.....	D	96	88			
Valves.....	D-M	88	45	Sn-Pb	5	
Wire and cable.....	M	31	13	Pb-Sn		
Copper-clad steel wire and BX	T	70	10		90	30
Tinsel, cordage, magnet wire...	M	50	20			
Insulated wire.....	M	91	30			

* M = Melt; T = Matte; D = Drill

TABLE 2.—*Rough Grouping of Metallic Scrap of Which Definite Proportions Should Be Taken (Class B)*

Copper and Lead-bearing Scrap	*Sample	Copper			Free Iron	
		Per Cent.		May Contain	Per Cent.	
		High	Low		High	Low
Ashes, all kinds.....	M-S	75	5	Pb		
Battery-plates.....	M	85	68	Pb-Sb		
Borings, all kinds.....	S-M	97	10	Sn	50	
Bronze powder.....	S	85	75	Sn		
Buffings, all kinds.....	S-M	65	9			
Carbon blocks and dust.....	M-S	79	69	Pb-Sn		
Cement copper.....	S	60	20		5	
Cobbings.....	M-S	25	15			
Concentrates, brass.....	M-S	70	5		10	
Copper cinders and clinkers....	S-M	30	7			
Copper oxide and scale.....	S	95	45			
Copper ingots and pigs.....	D-M	99	85	Sn		
Drosses (Cu, Pb, Sn, terne, solder, babbitt and die-cast- ing).....	M-S			Sn-Pb	20	
Dusts, copper.....	S	85	15			
Fuses.....	M	60	30			
Filings.....	S-M	75	65		40	
Grindings.....	S-M	75	10		20	
Jack-strips.....	M	45	10		50	40
Lamp-bases.....	M	30	5			
Lead oxide and mud.....	S	70 Pb	45 Pb	Pb		
Lead sulfate.....	S	65 Pb	35 Pb	Pb		
Mattes.....	S	60	20			
Mud, foundry, rumbling.....	S	25	3			
Residues.....	S	40	25			
Shells, shotgun, brass, copper..	M	70	25			
Shells, electrotype.....	M	50	30	Pb		
Screenings, siftings, sievings...	S	70	45			
Skimmings (aluminum, brass, zinc sal-ammoniac).....	M	73	7	Pb-Sn- Zn-Al		
Slags, all kinds.....	M-S	21	0	Sb-Pb-Sn		
Spatters.....	M	65	35			
Swcats.....	M			Sn-Pb-Sb		
Sweepings.....	S-M	65	25			
Tailings, separator, magnet...	S	56	7			
Turnings, all kinds.....	M-S	99	23	Sn		
Washings.....	S	85	4			

* M = Melt; T = Matte; D = Drill; S = Screen

The aim at all times should be to minimize the personal equation, but to keep in mind that free metal is invariably present and that the lot is probably very unhomogeneous, or rather, heterogeneous.

In Class A (Table 1) the representatives of both buyer and seller select a number of pieces, which will be as nearly typical as is consistent with the weight and value of the lot. Hand work is not only unavoidable, but desirable. For example, a carload of briquetted or bundled wire may have "bricks" or "bundles" consisting wholly or in part of heavy insulated and cotton-covered wire, soldered ends of clean wire, lead-covered cable, and copper gaskets. Haphazard selection is wrong and taking every tenth may be very unfair; a number of each type, dependent on occurrence, should be taken and one-half of each "brick" sheared off and cut up into small pieces. The pile can then be mixed with shovels and quartered down to a weight of 10,000 to 30,000 gm. (20 to 60 lb.) and melted.

Another example would be armatures, which run, of course, very high in iron. A sufficient number of each type, which may be one of each or more, may be first stripped by hand. A greater net weight can then be melted unhampered by unwieldy amounts of iron.

In tuyeres, one or two should be selected and the inside scraped and cleaned before drilling, if possible. Obviously, the tuyeres taken for a sample should be weighed before and after cleaning, although no melting is done.

It should be evident from the cases mentioned that the sample taken may be anything from 1 to 50 per cent. of the lot of scrap.

In Class B (Table 2), however, a more or less definite scheme can be outlined to cover a sample of 1, 5, 10 or 20 per cent., depending on the percentage of free metallics and the particle size. The main point lies in the manner of taking the sample. Each man engaged in unloading the car should be assigned to a certain area and the removal of the material should be along a definite line, say from right to left or vice-versa, setting aside the fifth or tenth shovelful for the sample. Large metallics should be picked out by hand, broken down, weighed and added in the proper ratio to the main portion. Certainly the men unloading should not be permitted to shovel here or there, taking a sample shovelful whenever or even wherever they may choose. The only justification for a percentage sample is in the regularity of taking it. A carefully taken grab sample is better than every fifth shovel taken haphazardly.

In all cases, half the final crude sample should be held as a reserve in case of dispute.

MOISTURE DETERMINATION

In general, the regular sample will suffice. If the material is sold with payment to be made on the original or "as received" basis, it is necessary to ascertain the moisture, if for no other reason than the proba-

bility that the reserve portion may either lose or gain moisture in storage, and, of course, it is always advisable to dry before charging into a crucible. If the sample is to be screened only, either the sample must be dried at 212° F. and the resultant assay run on the "dry basis" or else the sample may be burned with or without the addition of kerosene oil. In either case it may be calculated to the "original basis."

If material is sold with payment and treatment charges on a dry basis, a moisture test must be run. The usual practice is to take either the regular sample or a portion (weighing, say, 10,000 gm.) and dry either in an oven or by the furnace. The moisture determination, therefore, may be very important, for it may not only reduce the treatment charges through smaller weights but may throw the material into a higher schedule and consequent lower rate.

The moisture determination, in most cases, should be made by drying at a temperature about 212° F. Fine material is sometimes calcined, but while this shows a loss of other volatile and combustible matter as well as moisture, this loss may be partly offset by oxidation of finely divided metal. As a matter of fact, material like copper cements and some mattes will show first a loss, but finally a gain when dried at slightly over 212° F.

Copper oxide or mud from wire or cable may contain both water and oil or grease. The samples are generally first dried and then burnt. Such moisture determinations are often unsatisfactory.

ELIMINATING OIL AND GREASE

In the sale of turnings, etc., for remelting purposes, the matter of oil and grease is often an issue. A 5 or 10 per cent. sample is taken of thoroughly mixed material, coned, quartered and remixed and placed in a container, preferably a bottle. When sold to a smelter, this class usually calls for burning the sample. Before and after burning weights must be taken. When sold to a brass foundry the turnings should be washed, first with alcohol and then with ether, before analysis, the loss in weight being ascertained.

REMOVING IRON

Whenever possible, free iron should be removed with a magnet or by hand before melting. If the sample is one that admits of screening, it is best to pick out by hand the larger pieces and use the magnet only on the coarse scales or metallics. If used on the fines, values may be removed at the same time, especially if oily. If the iron is attached to the other metals, as in armatures, generators, bed-rods and the like, it is sometimes possible to sweat the material. When necessary to actually melt the other metals, the iron should be taken out by skimming; it should always be weighed and the weight included in the sampling data

given on the bag. Unless carelessly done, the "wash" or copper adhering to such iron can be disregarded. If the material cannot well be sweated or melted into a good bar, the whole sample should be melted with the addition of sufficient sulfur to form a clean low-grade matte. This is especially advisable on material containing more iron than copper and in copper-clad steel.

SIZE OF THE SAMPLE

The weight of the final sample is dependent on (1) the melting equipment available and (2) the size and character of the pieces in the shipment. Ore can always be broken down and crushed; scrap rarely. The larger the particle size, the fewer the particles, and consequently the larger the sample required. The more varying the character of the pieces, the larger the sample required; 100 lb. might be none too much to melt of an iron composition, while 20 lb. might be plenty for dirty grindings or No. 2 wire.

For convenience, it has been the habit at some smelters to take even and definite weights for melting, such as 50 or 100 lb. (10,000 to 20,000 gm.). This is not good practice, for to get an even weight, something must be taken from or added to the final sample. Either a definite quarter or two quarters should be weighed "as is," whether selected by the shovel or by hand. There is no doubt that taking round figures, such as multiples of 10 or 100, minimizes clerical errors, in one way, especially in calculating the final assay, but the weights of the various portions after melting or screening are rarely even, so the clerical and mathematical advantages claimed are at least open to question.

MELTING EQUIPMENT AND PRACTICE

Copper, brass, scrap and copper-bearing material are usually melted in graphite crucibles, sizes from No. 100 to 150 being in general use. On account of the bulky nature of some of the scrap, it is best to be generous as to capacity. It is sometimes necessary to charge in several batches and possibly add more flux.

The furnaces may be either of the pit or tilting type, and are usually oil-fired. (Fig. 1 shows furnaces used by United States Metals Refining Co. at Chrome, N. J.) One man can handle two furnaces, but two men are required for melts much over 50 lb. This increases sampling costs, which are, of course, passed on to the shipper, and increased accuracy gained by melting a larger sample may be more than offset by the increased sampling charges.

All the samples should be delivered to the melter in containers plainly marked with the lot number. The charge can be placed in the crucible and the necessary flux added. Borax is generally used alone or with additions of charcoal, soda ash, and sometimes sand, depending on the material. Salt should not be used with samples in which the copper and

tin are to be determined, and in melting all samples for lead, tin and antimony, care must be taken to avoid excessive heat in melting.

A crucible will give 12 to 14 melts, on the average. Clean copper-bearing material will give from 18 to 20 melts; very dirty and irony material from 5 to 10, and battery lead rarely over 7 or 8 melts. High-zinc and irony samples are the hardest on crucibles.

After the sample is properly melted the free iron may be skimmed off into a pan; tagged, cooled and weighed. The slag is then skimmed into a pan, or better still, poured with the metal into a mold, the slag broken loose and weighed. If very small in amount or of a light color, it can be disregarded altogether. This is usually the case in heavy and light

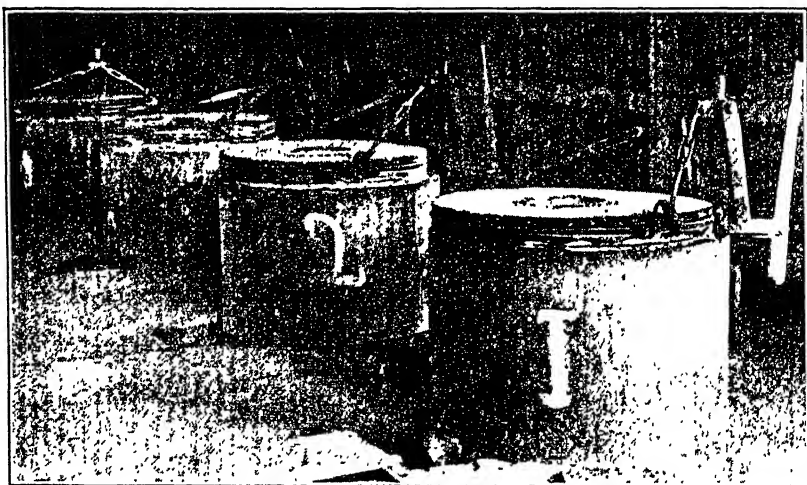


FIG. 1.—FURNACES USED BY THE UNITED STATES METALS REFINING CO., CHROME, N. J., FOR MELTING SAMPLES OF COPPER-BEARING MATERIAL.

copper, clean brass, etc. Slags from furnace products, dirty scrap, etc. should be weighed, crushed to 60-mesh and sieved.

Shot will often be found. Experience has shown that the copper assay of this shot will rarely vary more than 3 per cent. from copper assay of the bar, so that it is generally cheaper and sufficiently accurate to weigh the shot, deduct the weight from the slag and add the weight to the bar, thus saving an assay. The slag as ground is mixed, quartered and bagged. If it is necessary to remelt a bar because of hard irony shots, it is best to add more flux. The weight of the second slag should be added to that of the first, and the two should be mixed.

After skimming, the melt is poured into a flat slab or bar, preferably not over 1 in. thick. Drillings should be taken from all portions, drilling through each time. The drillings should be mixed carefully before bagging, but it is rarely necessary to grind them.

Great care must be taken in tagging and marking, and the stamping of the bar is especially important, for it is the heart of the sample.

The after-melting, or A. M., weight should be taken and noted; in fact, all weights pertaining to any sample should be checked by two men. This point cannot be stressed too much.

PREPARATION OF SAMPLES

After the sample has been selected by one or the other of the methods described, it may be prepared by one of several different schemes, depending on a number of conditions, governed (1) by the facilities at hand; (2) by the weight and approximate value of the lot; (3) by the size and character of the material. The methods at this stage of final preparation comprise melting, screening, and matting.

Melted Samples

Melting is the method generally used. The material, either dried or undried, may be charged directly into a graphite crucible or the fines may be screened out and only the coarse material melted, with or without prior removal of the iron.

Screened Samples

Material suitable for screening is usually dried first, for obvious reasons. It may also need grinding or crushing in a mill or possibly by a jaw-crusher and set of rolls. Sieves 30 to 60 mesh are used, dividing the samples into portions called fines and metallics, or pulp and scales.

It is generally advisable to assay each portion separately. The iron in the scales, shot or metallics should usually be removed with a magnet. No attempt should be made, however, to remove iron in this way from the fines or pulp portion.

It may be good practice to melt the coarse portion if it is sufficiently large and unhomogeneous. Before and after melting, weights should be taken. In melting, it may be advisable to add a small amount of borax, but it rarely is necessary to weigh or analyze the small amount of such slag.

Matted Samples

For iron material, this method, when it can be conveniently used, is best of all from the standpoint of accuracy. When properly carried out the resultant matte can be crushed, pulverized and thoroughly mixed. Only one determination of copper is necessary and the buyer's and seller's assays should and do check within 0.25 per cent. or less. It is peculiarly suited for iron material but the cost is high and very few smelters have facilities for getting a high enough heat, so that often a slag will contain free metal. This slag must be ground and separated into fines and shot, making three portions of the sample to be assayed.

The final result, though more nearly correct, may have increased the sampling costs beyond reason.

On the other hand, some samples have so much iron not readily removed except by melting that, despite the care used in skimming after sweating, the resultant bar contains hard spots, which easily break a drill. Such a bar must be remelted with additional flux, thus giving two slag weights and two A. M. weights for the bar, with a consequent increase in labor and fuel cost and still further chance for clerical errors and losses.

ALUMINUM SCRAP, DROSSES AND SKIMS

Aluminum scrap is rarely sold on joint sampling at the works of the buyer. One of the largest works handling this confines its purchases chiefly to material of which the origin and composition are known.

Drosses and skims, however, require some form of sampling, as they are sold on the basis of "metallic or free aluminum," no payment being made for the oxide. Up to recent times the practice has been to draw a sample by either the grab or shovel method, reduce in quantity, crush somewhat if necessary, and screen through a 60-mesh sieve, discarding the fines obtained and fluxing only the coarse.

The free metal having settled to the bottom of the crucible, the heat is continued a few moments and the crucible removed, cooled and broken. After cleaning, the button may be sawed for purposes of analysis. As a rule, only the aluminum is paid for, but the button may contain other metals present in the original alloys; copper, for instance.

Tests made in our laboratory with zinc chloride and calcium fluoride, the usual fluxes given out in the trade, indicated that the finely divided metal contained in the portion under 60-mesh was not recoverable. We have reason to believe that a change in the method of fluxing would partly avoid oxidation of particles as fine as 80 mesh.

BRASS AND COPPER SKIMMINGS

This material will usually contain metallics of appreciable size. It may be sampled by hand by the tenth shovel, or "one and nine," as it is sometimes called, or it may be passed through an ore-sampling mill. The latter is a distinct innovation in sampling practice, introduced at the copper refinery of the United States Metals Refining Co. at Chrome, N. J. The mill is used only for carload lots.

As in hand sampling, large metallics should be picked out and broken with a sledge. If there are many of these, they should be weighed and a proportionate amount added to the main sample as delivered from the last cutout used in the mill; if few, they may be broken up and thrown in with the rest of the material. The mill is equipped with four cutouts (any one or all can be used), each giving a 20 per cent. cut. If the particles are of sufficient size, the material may be passed through a jaw-

crusher, gyratory crusher and three sets of rolls in the order given. The full use of the cutouts will give a sample equivalent to $\frac{1}{625}$ of the lot with a weight ranging from 150 to 250 lb. Depending on the setting of the final rolls, the maximum particle size will vary from 0.25 to 1 in. This sample may then be passed through a laboratory-size crusher, quartered once or twice as needed and melted to a bar and slag.

Ashes, cinders, clinkers, cobbing, concentrates, drosses, muds, residues, slags, fine spatters, sweepings, tailings and washings are sampled in a like manner except that they need no crushing, as they contain only fine material. They should be screened and assayed "as received" and not melted.

LEAD DROSSES AND SKIMMINGS

Waste of this type may be sold on the "metallic button" or total lead content basis. In either case, after a sample is obtained by the shovel method, it is melted down under a reducing flux.

If the contract calls for "metallic," most of the lead, tin and antimony will be in the button, together with some of the impurities such as copper, arsenic, etc. If "total lead" is called for, both the button and slag are assayed and the flux used is of a more simple nature, no care being taken in reducing other than to get a clean slag, which may contain as high as 60 per cent. lead. In the "metallic" melt, the flux is highly reducing, the losses of metal retained in the slag being *partly* compensated for by the somewhat impure button thus obtained.

BATTERY LEAD OR PLATES

This consists of old wet batteries of the automobile and radio type, and the lead plus antimony will vary from 70 to 85 per cent. The antimony will run from 3 to 4 per cent. as a rule. The container is usually not shipped and there is a marked tendency to cut off the lugs and sell separately. There is always more or less dirt but in the past most of the trouble has been from the moisture. Some have dried at 100° C. (212° F.) but the better practice today is to bake the sample.

On an average, the shipment will consist of lugs, connectors, plates, oxide and sulfate from the plates, and wooden or rubber separators. The separators will make up from 7 to 10 per cent. of the wet mass and will show a moisture about 10 times as high as the plates and lugs combined. The lugs will assay about 96 to 98 per cent. lead and antimony. An average lot of battery plates should contain around 8 per cent. lugs; a lot may have none; if mostly whole batteries, it might show as high as 15 per cent.

The plates and oxide, then, make up the bulk of the material and the difficulty in sampling is to get a proper ratio of plates, lugs and separators; or liners, as the last are sometimes called. The proper procedure is to

take every fifth or tenth shovel, as the case may be, and when the sample is drawn have it weighed. Break and cut off the lugs and weigh to find out the percentage of lugs in the material. Break up and crush the plates and liners, mix and quarter down to approximately 100 lb. Then combine the proper proportions of lugs and plates for the melt sample. Of course, a reserve sample should be held and moisture run at a temperature high enough to drive off moisture and other volatile matter but not high enough to melt or sinter the sample.

The smaller the sample melted the more care must be taken, although if brought to a fine state of division a portion as low as 100 gm. may be melted in a clay crucible; the latter cooled and the button broken out and weighed. This practice is followed at the works of one of the large smelters and is probably the cheapest in labor, fuel and material.

Another works, which buys on metallic yield, melts from 250 to 350 lb. in a large kettle. The slag must be skimmed off and the metal ladled out into bars. The cost is high and the time long.

Trade practice in general calls for "metallic yield," "metallic button," "metallic recovery" or "metallic" (all four terms being synonymous) as the final operation of determining the value of battery lead and similar material; and there is little doubt that when properly carried out the results may be sufficiently near the truth. As mentioned before, the button is weighed and is assumed to be $Pb + Sb$. The slag is assumed to contain no more $Pb + Sb$ than will be lost by the smelter in refining and is, therefore, disregarded in this method. Ordinarily then, there will be no necessity for analyzing the button or the slag. As a matter of knowledge, a 300-lb. sample, typical of the practice at one works purchasing on the metallic basis, was thus melted and assayed. The metallic, or button, was 78.10 per cent.; the slag amounted to 6.78 per cent. of the sample. The partial analyses of each and the calculations showing the comparative and composite values is given in Table 3.

Our experience is that on a large melt, the sample is more representative but the metallic yield is too low because of the character and amount of slag. Conversely, on a small melt, it is more difficult to obtain proper proportions of lugs, plates, liners and oxide but the slag is more apt to be relatively small and clean.

In another experiment in our laboratories, the total of the actual $Pb + Sb$ was 0.50 per cent. more than the metallic button run down on a large portion of the same sample.

$Pb + Sb = 83.00$ per cent. vs. metallic button, 82.50 per cent.

The slag on the first sample was high in both lead and antimony. The slag on the second, as usual, was not saved and analyzed.

The better practice, therefore, in the opinion of our organization, is to melt from 50 to 100 lb. in a large crucible to a clean slag free from shot.

TABLE 3.—*Partial Analysis of Typical 300-lb. Sample*

	Lead, Per Cent.	Antimony, Per Cent.	Tin, Per Cent.	Arsenic, Per Cent.	Copper, Per Cent.	Iron, Per Cent.
Analysis						
Bar.....	94.54	4.85	0.23	0.10	0.00	0.27
Slag.....	39.57	2.93	1.01	1.33	1.75	4.28
Reconciliation						
Bar (78.10 per cent.)...	73.83	3.79	0.18	0.08	0.00	0.21
Slag (6.78 per cent.)...	2.68	0.20	0.07	0.09	0.12	0.29
Sample.....	76.51	3.99	0.25	0.17	0.12	0.50

	PER CENT.
Metallic yield — lead + antimony + impurities.....	78.10
Lead plus antimony as assayed.....	80.50
Difference.....	2.40

The slag is skimmed and crushed. Either the melt is poured, the slag broken off and the bars sawed, or a dip sample of the metal is taken and poured into a small bar, which may be rasped down. The weights of both metal and slag are taken and both portions assayed for lead and antimony, payment being made on that basis, less the usual deduction and with a premium or penalty for plus or minus 80 per cent. lead plus antimony.

Baking losses will run as high as 15 per cent., called "moisture," but most contracts specify or allow 1 per cent. The average is 5 to 7 per cent.

TIN AND TERNE DROSSES, SKIMS AND ASHES

Tin drosses are often sold on the basis of "metallic button" or "recovery." They may contain as high as 35 per cent. moisture and much zinc and ammonium chlorides. In this wet condition they are in the form of a thick sludge and the sample is best taken while barrels are being loaded. A sample of 10 per cent. or 5 per cent. should be sufficient. (Another method of drawing is to pour off the water, mix and take a pipe sample of the sludge.) After mixing thoroughly the sample is placed in two-quart glass jars, covered tightly, and both barrels and sample are weighed. The sample is then sent to the laboratory and washed thoroughly by decantation, dried and weighed. It may be then passed through a set of laboratory rolls, sieved through a 40-mesh sieve and the metallics or scales weighed. The pulp or fines is best taken by difference. If sold on "metallic button" by the "coal" method, disagreements are minimized by making a joint assay in the presence of both buyer's and seller's representatives.

Coal Method

Proportionate amounts of fines and scales, together with a flux made up of soda and hard-coal dust, are mixed in a small crucible. Four times as much flux as sample is taken and the latter may be from 100 to as much as 400 gm. Triplicates are made up and melted in a gas, coke or oil furnace. The melt, when not cooled in the crucible, is preferably poured into a flat bar rather than a button. If it is to be analyzed, raspings are better than sawings. Segregation in bars, from drosses containing antimony and lead, is marked, so the raspings should be thoroughly mixed. As a matter of fact, at least two melts should be taken and averaged. The slag is ignored. Results are low but are claimed to be in line with smelter recovery. As much as 4 to 15 per cent. of tin may be lost.

Cyanide Fusion

The cyanide fusion is less often used to ascertain the metallic button and is carried out in a manner similar to the old determination in tin ores, using twice as much cyanide as sample. It is not suitable for terne drosses. A more accurate method is to screen the sample through a 40-mesh screen after reducing. A cyanide fusion on 10 gm. in triplicate will give a button on the fines, which may then be rasped down.

If much iron is present the button will have hard spots and care must be taken again to avoid segregation. The coarse screenings will be of a scaly character and usually brittle. They may be further cut and broken down, riffled or melted under a flux or with resin, cast into a bar, weighed and rasped or sawed. Correction must then be made for the melting loss.

Iron is an impurity and may be penalized. If present in sufficient quantity the determination can be made only by a wet method on the original material. Lead is also best assayed in a similar way.

Most Accurate Method

The most accurate method of all for such material is the one employed at the Balbach Works of the United States Metals Refining Co. on solder drosses, sweated radiator dross. This consists of melting 20 to 60 lb. of a suitably prepared sample with soda and charcoal, in a graphite crucible or large ladle. The slag is crushed and ground and the bar may be sawed lengthwise and one-half rasped down. Both bar and slag are assayed.

ZINC ASHES, DROSSES AND SKIMMINGS

These may be very wet and full of salts such as zinc and ammonium chloride or they may be dry with practically no chloride. They are sold on "sample," the latter being often but a grab wrapped in paper, whether

wet or dry. Determination is always required for the total zinc content and often for the moisture and chlorine, the latter being penalized. In sampling, care should be taken to obtain the proper proportion of metallics and fines. The sample is best bottled.

LAMP BASES

This material consists of the scrap from electric-light bulbs. The glass from the bulb itself is usually present in only a small amount, but that from the inner tube in which the wires are sealed will be present in varying amounts in the shipment. A base will consist of four parts, the ratio by weight of each part being as follows:

	STANDARD TYPE 1 IN. DIAM.	LARGE TYPE 1½ IN. DIAM.
	PERCENTAGE BY WEIGHT	
Brass shell.....	20	25
Wax or plaster.....	49	37
Glass tube.....	30	37
Lead-in wires.....	1	1

The material is sometimes shipped "as is." The copper content will vary from 5 to 30 per cent., although the brass assays 66+ per cent. copper.

Some shippers crush to remove as much glass and wax as possible, and such lots may assay as high as 50 per cent.

ELECTROTYPE SHELLS

Often sweated before shipping, they still run high in lead, and in the past payment was not, as a rule, made for that metal. They are now at times sold for both copper and lead contents and must be sampled on that basis. On casting, the two metals will separate in two layers. The lead portion will be fairly pure lead; the copper portion will contain more or less lead and, of course, impurities.

LEAD-COVERED CABLE

Whether called "lead-covered cable" or "copper-filled cable lead," payment is made for the actual copper and for the lead plus antimony.

The cable is best stripped by hand or machine and the lead and wire portions melted separately. The former will show a low melting loss, the A. M. weight being taken as the Pb + Sb + Sn.

The copper portion, being stranded wire covered with insulation and coated with tar or similar material, will give a high melting loss. Shippers cannot understand why payment is not made for copper on the A. M. weight, as on the lead; obviously, copper melted and cast into a sample bar will contain considerable oxygen, to say nothing of other metallic impurities.

CARBON BLOCKS

Carbon blocks are made usually in four copper types ranging from 70 to 85 per cent. The "pig-tails," or stranded wire sometimes attached, will assay around 96 per cent. Almost all of the blocks and dust will show appreciable amounts of lead and possibly tin. They will assay from 69 to 79 per cent., rarely over 75 per cent.

They must be melted with a flux containing niter, to oxidize the carbonaceous matter, or losses in melting are apt to be high, on account of dusting.

This material may be ground in a jaw-crusher and roll-crusher but this is not advised, because, unless extraordinary care is taken in cleaning the apparatus, it might result in salting some other samples. The fine powder is not necessarily dusty of itself.

BRONZE POWDER

This material is so fluffy that it causes a great amount of trouble. It flies about and sticks to everything and should not be handled near other lower grade stuff.

AUTOMOBILE RADIATORS

Automobile radiators are not sampled but are sold on weight less iron, which is in the water connections, plates, strips, etc. Some refineries have their own list of iron deductions for radiators, as does The National Association of Waste Material Dealers. The iron deduction will range from 2 to 20 lb., the latter from truck radiators. Fords will show from 2 to 9 lb.

The remainder is chiefly copper, brass and dirt. The solder will average about 7 per cent. The recovered solder may contain as high as 40 per cent. tin. Some dealers dismantle and ship the cores; others sweat to recover the bulk of the solder. The tendency at present is to use less tin than formerly in the solder, so the figure given for that metal will probably be too high for the future.

JEWELRY AND DENTAL WASTE

An entire paper could be devoted to this important and valuable material but only the phase that affects the larger smelter will be taken up here. There are innumerable establishments making jewelry and many small dental laboratories spread over the country. The waste from the floors, sinks, benches and furnaces, etc., is saved and collected by small dealers who are usually called "refiners." Combustible matter is burnt by them to ashes, which are called "sweeps."

Metallic waste, such as clippings, punchings or stampings, is usually cast into bars called "bullion." Other names are "platers' scrap" and "filled clippings." Only the low-grade type of bullion free from the platinum group will be discussed.

SWEEPS

This material is always shipped in barrels or drums, usually sealed. Shipments vary from 1 to 7 bbl., rarely more. The entire lot is spread out on a clean floor and thoroughly mixed by shoveling, coning and quartering. It is often hygroscopic, because of the presence of sodium salts from cleaning solutions, and the moisture will run from 0.3 to 1.5 per cent. A sample for moisture is best taken at the time of dumping the barrels and immediately after weighing. This sample is usually dried at the works in the presence of the shipper's representative. As a steam bath is used, about 0.10 to 0.15 per cent. of moisture is usually left. The main lot having been quartered down carefully to about 12 lb. maximum, the sample, consisting of two opposite quarters (the other two being held as a reserve), may be placed in a pebble mill or bucked down by hand on a bucking-board and sieved through an 80 to 120-mesh sieve, preferably the latter, to remove the scales or metallics. These will rarely run over 2 per cent., as in most cases the shipper has already sieved the sweeps through an 18 to 60-mesh screen before shipping.

The sample should be weighed *before grinding* and the metallics weighed *after screening*. The fines weight is then taken by difference. On account of dusting losses, an improper ratio would be obtained if this were not done. The fines or pulp is then mixed, quartered and bottled.

The metallics are unhomogeneous in value and particle size and, being small in amount, should be scorified with an excess of lead to 20 to 100 gm. poured in a small button or slab. The slag is cleaned and the button rasped, quartered and divided into three portions.

Fines and metallics are assayed separately for precious-metal contents only. No allowance is made for base metals.

Some works, instead of taking the struck weight of two quarters, weigh out, grind and sieve definite weights such as 16, 20, 30 or 40 assay tons. This is not recommended, for while it is convenient for calculating, it is wrong in principle.

Watch-case Bullion or Platers' Scrap

This material is very valuable. Silver will run from 1 to 50 per cent. gold from 2 to 15 per cent. and copper from 50 to 80 per cent. Payment is made for all three, the balance of the alloy being chiefly zinc and nickel.

It is shipped generally in bars, the shipments rarely exceeding 13,000 oz., as that is the full capacity of the crucible used for melting. The entire shipment, after weighing, is melted with a little borax and charcoal to a clean melt. The slag is skimmed off and poured into water to remove the bulk of the remaining charcoal, then dried and reserved.

The molten mass is stirred vigorously by plunging an iron ladle up and down with a twisting motion. A small hot clay crucible is then

dipped into the metal and a shot sample taken by pouring into a pail of water in which a wooden board rests. The hot metal is thus spattered under the water, the object being to make as little "matted" shot as possible. The very fine shot should be screened out after drying and the very coarse picked out, only the medium size, about 18 to 40 mesh, being used at one refinery, and $\frac{1}{8}$ to $\frac{1}{4}$ in. at another. The coarse assays the same but is awkward to handle at the balance: the fines will carry any slag or dirt inadvertently carried over from the surface of the melt, and should not be used for assay, as it also has a greater tendency to oxidize, by reason of the zinc present.

The metal is ladled out into bars and weighed, and the slag reserved as described above, is thrown into the crucible. The heat is raised and the pot sweated. The metal recovered is poured out, cleaned of charcoal and the total metal recovery weighed. The assay is figured on this after-melting, or A. M., weight. The treatment charges may be based on the before-melting, or B. M., weight.

A long series of experiments initiated by this laboratory and carried out some years ago at one of the refineries proved that, even on the highest grade material, no segregation troubles occurred provided the melt was thoroughly mixed, as described above. It was feared that the combination of high gold, zinc and a high-melting metal—nickel—might result in an erroneous figure if the sample was dipped from near the top. The assays of samples taken at intervals from the bottom to the top by this method showed accordant results.

Small bars, however, weighing only a few hundred ounces or less may be drilled in six holes diagonally across the bar, drilling through each time. It is very difficult to pour clean enough on a small weight to obtain a true A. M. weight, so a melted or shot sample is not always taken. The last method gives a more homogeneous sample, however. As these small bars often contain large amounts of platinum metals the relative accuracy of the two methods should be ascertained by a series of carefully controlled experiments on actual shipments.

ACKNOWLEDGMENT

The author wishes to express his thanks for the hearty cooperation extended to him by many while he was preparing this paper.

DISCUSSION

T. A. WRIGHT.—On page 650, I want to add one point: Often when drillings are taken from ingots such as are cast from melted samples, they are not ground, and there is ordinarily no necessity for it, but in large and carload shipments, handling valves, ingots and things of that kind, the samples should be ground or the results will be indeterminate. Great care must be taken in tagging and marking, and stamping the bars is especially important; that is, in relation to the melted samples. I suppose at

least 75 per cent. of the samples taken on copper scrap and on the other metals are melted. The weights before and after melting are very important and should be checked. More trouble is caused by clerical errors than by anything else.

The preparation of the samples is the all-important thing, and this is going further than what I said in regard to size of sample. No matter what scheme is used, it is governed by these factors, by the facilities that are at hand, by the weight and approximate value of the lot, and by the size and the character of the material; so that really such methods as may be given here are merely outlines of practice which must be modified many times to meet the particular situation that has developed.

W. J. BROWN, Brooklyn, N. Y.—Mr. Wright has particularly pointed out that the complexity of secondary metals makes the taking of the sample a matter of vital importance to the chemist. The work of the chemist is of little value if his sample is not representative of the original material. To leave the taking of a sample to personal selection makes sampling simply a matter of judgment. Unfortunately, many materials are of such a nature as to render other methods impractical.

When in the process of quartering, large pieces are broken down to proper particle size, returned to the sample, mixed and quartered, it is important that proper particle size be controlled by the amount of material being handled and that the quantity of sample submitted to the laboratory be controlled by particle size.

Separating into several parts either by sieving or by selection with the consequent weighing of each part increases handling charges. Dividing the "assay" into parts such as a bar and slag increases the laboratory cost. Should tin, lead and antimony be desired, it is necessary not alone to determine those metals but also copper, iron and arsenic, which may be present as impurities, and increasing the number of parts multiplies the analytical work to that extent. When the lot represents a small shipment, sampling and laboratory costs make an item to be reckoned with.

The many varieties of scrap on the market and the complexities of their compositions call for the exercise of judgment on the part of both sampler and chemist. Mr. Wright has made a valuable contribution to the secondary non-ferrous metals field in pointing out the difficulties of handling this class of material.

E. H. LAWS, New York, N. Y. (written discussion).—Mr. Wright is to be congratulated on the writing of a valuable paper on a very difficult subject.

In connection with the sampling of copper-bearing scrap material, I submit the following notes on possible improvements in practice:

It is evident that copper-bearing scrap material is most difficult to sample, owing to the variation in the value of individual particles and the high melting point of the metallic pieces. Assuming a shipment which is representative of most of the material received, it will contain metallics varying in value from pure copper to metal of no value, or even to a minus value to the copper refiner. The difficulty in sampling is increased by the variation in size of the metallic parts and because of the presence of moisture and combustible matter, such as grease, dyestuff residue, crude oil, sawdust, paper and rubber. In order to get accurate results it is obviously necessary to cut out large amounts for samples and these notes apply to the handling of such large samples, which may reasonably vary in weight between 100 and 1000 lb., according to the size and character of the shipments.

1. *Drying Tests*.—With steam-heated compartment with some controlled circulation of air and with special trays or buggies, it is practicable to dry large samples at nominal expense. This has been done with 1000-lb. ore samples in lead-smelting practice in order to speed up and make more accurate the further reduction in size to laboratory pulps, and with most successful results. In determining the value of a shipment, the moisture content is of prime importance. If a large sample is taken,

the moisture determination made can be relied on to a far greater extent than that made on the usual small grab sample. The determination should be checked by further drying the sample of the fines as it is cut down for the laboratory. In addition, the material is put in proper condition for further sampling operations. No expenditure for sampling can be made which will yield more benefit and valuable information per dollar spent than that for drying large samples. The results are convincing to both buyer and seller.

2. *Burning Tests.*—Nearly all scrap-metal material contains some combustible matter. For example, in dyestuff residues, the combined moisture and combustible matter may reach 66 per cent. It is evident that an accurate determination of the combustible matter or moisture plus combustible matter may be of greater importance in showing what the material is worth. Drying puts the sample in good condition for burning and these tests can be carried on best in a small cast-iron or steel kettle, with proper setting and with hood for taking away the smoke and fumes of grease and acid. There should be provided convenient storage for samples and floor space for cutting down the fines produced. With a small battery of such sampling kettles, the operating cost, which is chiefly for labor, can be kept at a very reasonable figure. It is apparent that the cost for floor space and equipment is low.

3. *Rejection of Worthless Metal.*—Any lead alloys or other metal fusible at low heat which separate in the kettle can be segregated. Any worthless metal which can be detected by inspection can be picked out. Scrap iron can be removed with a magnet. A hand electromagnet with control switch is suitable for this work. It will operate in the same manner as a large lifting magnet and save expensive labor, besides making a better separation.

4. *Grinding.*—After preparation of a sample as suggested, it is in first-class condition for grinding preparatory to separation of coarse metallic pieces from oxides and other earthy material which can be ground fine. For this grinding a ball mill is indicated and has been tried out. The design must be special in that it must be possible to easily discharge the entire contents of the mill. The metallics, of course, come out clean and burnished and the fines ground to practically any fineness desired. The separation between coarse metallics and fines is sharp and after passing a suitable screen to hold back metallics, the fines can be cut down directly to laboratory pulps.

5. *Melting.*—If the metallics obtained are too large in amount for a convenient melt, they can be fairly divided by quartering, after shearing the large pieces. It is evident that the melting operation will be quite different with clean copper, brass and bronze from that in which oxides, dirt, scrap iron and other miscellaneous material must be fluxed or matted. Probably the best furnace for melting is the ordinary coke-wind furnace with fan blast and ample stack, which was used by the Nichols Copper Co. at Linndale. The high-frequency induction electric furnace, using crucibles, would be suitable because of speed and good working conditions, but the cost of the power equipment for a furnace to handle 50-lb. samples is rather high at present. In connection with melting, it should be remembered that while the metallics can run to 100 per cent. of the sample, they ordinarily form a very much smaller percentage. Also, there is no necessity for melting, for sampling operations, any material that can be crushed fine.

All of the procedure, together with the recording of the various weights, can be carried on in the works as matters of regular routine and with accuracy. The sample mill furnishes a clear record of the results on each lot and this, together with an inspection report on the physical character of the material, gives a great deal of information before assays are made. This information is often of the greatest advantage in checking back.

The sampling of lead-bearing scrap is similar to that of copper-bearing material, but the melting operation is easier; in fact, no crucible work is necessary, nor is it

advisable. I believe that sweating to metal and dross is preferable. The necessity of working with large samples is equally great, because of the varying character of the materials in one lot. The moisture content is extremely important; for example, the moisture and acid in scrap battery plates can easily range between 2.5 and 12.5 per cent.

The best method in my opinion for sampling scrap battery plates is that developed by the Pennsylvania Smelting Co. Briefly, in unloading, each fifth or tenth shovel is reserved for sample, and this amount is further reduced by reserving each second or third shovel, making the final sample 1000 to 1200 lb. The net wet weight of this sample is taken and it is stored until it can be worked. On treating in a small kettle, the moisture and acid can be expelled, the separators, and other combustible matter burned, rejected metal picked out, the metal melted down and the fines (mostly lead oxides and sulfates) skimmed off without being slagged. The metal is cast into pigs, after a sample bar has been taken. The fines are coned and quartered to 40 lb., which goes to a copper mill, is riffled and the small sample sent to a disk grinder. These operations separate the large sample, at small cost, into metal, rejected metal and fines and determine the loss. The metal and fines are put into shape for accurate sampling and, with all weights available, composites representative of the lot can be made up in the laboratory for metal determinations.

E. FITZPATRICK, New York, N. Y.—Mr. Wright says it is not necessary to take a sample of the slag from the clean metal melts, but we have found that we are taking a great chance if we discard any slag. When we first started melting these scraps, we did discard all the slag, but several times we threw away slag that afterwards was proved to contain 3 or 4 per cent. of copper. In the last few years, we have sampled and analyzed every slag.

I agree with Mr. Wright that it is very much safer to grind the samples; there are likely to be some pieces of foreign metal present, such as steel or perhaps some lead. A 20-gm. sample cannot be homogeneous unless it is ground to pass a 20-mesh screen. That is our practice.

Of course, theoretically, mill sampling of ashes and material of that kind is the best way, but we have proved that hand sampling is equally good; in fact, in some ways it is really better, because the taking of samples can be watched from the beginning to the end and there is no chance of contamination. We made some tests last year on brass ashes and skimmings. We took three of each, and, as a regular sample, selected every tenth shovelful; then as we went along we took every fifth shovelful and called that the duplicate sample. In working those samples down all the way through, they checked within 0.3 to 0.4 per cent. on the average. That satisfied us very well on the hand sampling.

One objection to the mill sampling is that the mill must be thoroughly cleaned after each milling, as there may be a large metallic portion of rich material in some of the corners which might be picked up in a later procedure.

C. S. WITHERELL, New York, N. Y.—Referring to the seventh paragraph of the chapter on Melting Equipment and Practice, wherein the author says that "the drillings should be mixed carefully before bagging, but it is rarely necessary to grind them," I wish to partly confirm and elaborate upon Mr. Fitzpatrick's remarks.

If the metal is not too hard and the A. M. bar is flat and not too thick, a sawdust sample, which is excellent, can be easily made; such sample seldom holds together as unground drillings frequently do, consequently a thorough mixture is easily obtainable without much extra preparation. The saw cuts should be equally spaded throughout the length of the bar and should extend exactly halfway across if it is desired to keep the A. M. bar intact. For lead and equally soft metals a hand sample-punch may be

used, in which event the punch should be driven an equal number of times from top and bottom of bar.

Drilling is the favorite, fastest and most widely applicable method for sampling bars, pigs, cakes, etc., of metal harder than lead, but frequently, if the metal lacks brittleness, the drillings appear as spiral shavings which entangle with each other and thus prevent adequate mixing; under this condition grinding has decided advantages.

As the author states, drillings should be taken from all portions of the bar; to insure this it is well to place the drillholes according to template, preferably with "checker-board" layout. This procedure is particularly advised for A. M. bars derived from non-alloying mixtures of metals which, consequently, are far from uniform in composition. A favorite size of drill is about $\frac{1}{2}$ in.; larger sizes are slower and smaller sizes break easily.

Grinding of copper-bullion drill samples is standard practice at all copper refineries. It is common to use a small grinder of disk or cone type, with chilled toothed grinding surfaces designed to tear the product with little or no attrition when set sufficiently coarse. Usually the ground shavings are first produced in flake form, but, at least in one make of mill (the "Hance" mill handled by The J. H. Day Co.) the flakes can be rolled up into fine shot by simply running the mill backward for the final pass. This will make a thoroughly mixed sample of such nature that the whole quantity ground can be accurately cut down and divided for the retained (bagged) samples, and also insures that the subsequent sample weighed out for laboratory determination is entirely representative of the whole drill sample produced.

Due to the extreme segregation of the precious metals in copper bullion, it is common to grind the whole or aliquot part of the drill sample until all passes through a specified mesh (usually 20 mesh) and then to separate the fines that pass through a specified finer mesh (40 to 60); the coarse and fine parts are weighed separately and then may be proportioned accordingly on the analytical scale pan or may be assayed separately and the combined assay calculated. With metal devoid of precious metals, it would seldom be warranted to pass all of the retained sample through a specified mesh as described by Mr. Fitzpatrick; but if the A. M. bar contains slag or other friable matter it may be well to screen out the fines and handle in the same manner as is commonly done with copper bullion.

A mill of the type mentioned is not costly and it requires only a few minutes and very little additional expense to put the drillings through the mill two or three times; the thorough mixing resulting therefrom would of itself warrant the extra manipulation.

T. A. WRIGHT (written discussion).—The word "rarely," to which Mr. Witherell alludes, was poorly chosen; although more of such drillings have been assayed "as is" than as ground. The author agrees with the principle: the thought in mind was that many lots of scrap weigh from $\frac{1}{2}$ to 3 tons, which is a small amount on which to spend time or money that may not be justified. However, money spent in the preparation of samples may be returned with interest in the laboratory.

F. F. COLCORD, New York, N. Y.—Are the samples always coned and quartered down?

E. FITZPATRICK.—Always.

F. F. COLCORD.—Has the ridge method ever been tried instead of coning and quartering? In coning and quartering the sample, there is always the question as to whether the right amount of fines fall evenly throughout the cone and especially when the cone is worked down. Do you pull the apex one way or another?

Years ago in the sampling of the silver ores from the Cobalt district, there was a difficulty in that much of the value was in rather coarse metallics. It was necessary

to find a means that would prevent the throwing of the apex of the cone in one or another quarter. The so-called ridge method was therefore devised.

The sample might be anything to start with; say a fifth of the original ore was taken out by shovel sample and piled in a ridge, then, starting at one end of the ridge, the material was shoveled to the right and to the left, forming two ridges. These ridges were thrown back into a single ridge, then one shovelful was thrown to the right and one to the left until the sample was halved. One of the halves was divided by the same method, giving two quarter samples.

E. FITZPATRICK.—I am afraid the metallic would be too coarse.

T. A. WRIGHT.—Secondary metals today are sold on a very narrow margin of profit and handled, as I see it and well believe, by the refinery on a very narrow margin, so that we try at all times to keep the sampling at a minimum cost. The slag question depends somewhat on the difference in practice; one refinery will add considerable flux, another will not. If an ordinary No. 2 wire or No. 2 copper, heavy, clean brass, or something like that, is taken to a copper refinery, because it has aluminum in it and the brass foundry will not take it, our experience has been that the amount of slag is very small.

I know the Nichols Copper Co. is very particular about the slag, and I have often wondered why the expense seemed justified, because I have seen cases in which the correction in the analysis was 0.07 and 0.08 of a per cent.; within the limits—far below the limits, at which a split could be made.

In regard to grinding the drillings, I had in mind there also our experiments over a long period of time in the higher grade material in which we have been running 2-gm. portions against 40-gm. and finding excellent checks.

As for the mill sample, I think that is something for the future to bring out as to its efficiency. I am mentioning it, but not endorsing it. I think it has possibilities, yet I think there are faults in regard to cleaning which complicate it. It is absolutely necessary for the large metallics to be taken out before the sample goes to the conveyor.

C. O. THIEME, Detroit, Mich.—What is done in the refining, or rather in the analysis of terne plates and tin-plate drosses, particularly wet dross of which a sample is sent to the laboratories and washed through by decantation? How does the umpire or the referee analyst handle the soluble losses of the tin and how does he account for it?

T. A. WRIGHT.—That particular material is one of the things we dread to see come into the laboratory, and all too often we receive only a small sample, an 8-oz. bottle, or something like that. We try to get it down to a 2-qt. jar, which is mixed up in a sort of rule-of-thumb way in a big crock, stirred, and allowed to settle and decanted off. It must be evaporated down, which is a very tedious operation. We do not find much tin in the solution, as a rule, but tests should be made for it.

Significance of the Simple Steel Analysis

BY HENRY D. HIBBARD,* PLAINFIELD, N. J.

(Henry Marion Howe Memorial Lecture†)

AT THE beginning of a Henry M. Howe lecture it seems fitting to refer to Howe's great contributions to steel metallurgy, and particularly to the literature thereof. Most of my predecessors in this role have been able, from their association with him, to give direct testimony as to his great services; but although connected with him for years in a business way, I am less familiar with what might be called his academic side, except as it is embodied in his utterances, written and oral. Howe was adept in his genial way in extracting information in any line which, at the time, was of interest to him.

In a technical lecture one expects as a rule new knowledge—something about recent advances relating to the subject. On this occasion, however, I will ask you to go back with me to the primary department of steel lore, as the title of my lecture shows, to review the more fundamental features in that field for the purpose of seeking some gaps which we can fill and so give the subject renewed interest. In thus going back one must inevitably repeat many familiar facts, but that seems to be necessary in order to reach a clear understanding of the unrecorded matters.

The full interpretation of a steel analysis is at present unattainable, though much progress toward that end has been made. The ultimate analysis, giving the percentages of the five commonly determined elements, does indeed usually give a fair idea of what the steel is, and is ordinarily enough, with knowledge of its history and physical condition, to enable it to be assigned to some purpose or use for which it is reasonably suitable. But discrepancies arise.

Every maker and every user of steel knows, from at least occasional examples in his own experience, that the ultimate analysis is insufficient by itself to enable one to tell positively, from its interpretation, either the mechanical properties of the steel or its adaptability to the purpose for which it was made. Further, if, while having the specified composition, it is abnormal or unsatisfactory, why it is so. Good, bad and indifferent steels may have substantially the same ultimate analysis.

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† Presented at New York meeting, February, 1928.

The inconsistency of being better than expected, which means higher tensility or ductility or endurance in resisting shock or fatigue, some or all, comes as a rule from exceptional purity or freedom from harmful, or poisonous non-ferrous but undetermined ingredients. The inconsistency of being worse than the composition seemingly warrants may be due to one or more of a great number of physical and chemical causes which are included among the ills of steel. Some of these matters we are now to consider.

The crucible-steel maker of the last century exclaimed, "If you put the devil in the pot, the devil will come out of it." He had some justification for his opinion, as practically no oxidation or purification took place in the crucible, and the quality of the product depended chiefly on the purity of the melting stock used. Now, what is put into a steel charge, particularly in a basic furnace, while it must be suitable within limits, determines only in part what it is to be, and what service it is suitable for.

The commonest case of change in properties without change in composition is, of course, that of the ingot and the bar rolled therefrom. Heating and hot working improve the mechanical properties of practically all steels without appreciably changing the ultimate analysis. Up to certain limits, the more the work, the greater the improvement.

On this occasion we will limit our attention to plain or simple untreated steels and chiefly to those made by the open-hearth process. When bessemer, crucible or electric steels are referred to, the fact will be stated. Alloy steels, as well as heat-treated and cold-worked steels with their vast number of variations, must be reserved for the future. The upper limits of the five elements are taken to be: carbon, 1.5 per cent.; manganese, 1 per cent.; silicon, 0.6 per cent.; sulfur, 0.15 per cent.; phosphorus, 0.12 per cent.

VALUE OF ULTIMATE ANALYSIS

How far short the ultimate analysis may fall of telling the mechanical properties, a glance at Table 1 will show. This table gives approximately the maximum effects on tensility which may be and are effected by carbon, heat treatment and cold working, and shows the great extent of the field occupied even by simple steels. As a standard for comparison, Swedish wrought iron, considered as the purest iron obtainable, and having a tensility of 45,000 lb. per sq. in., is taken. The analysis is the same for all excepting the iron, but the variations in strength are exceedingly great.

By a full interpretation of an analysis is meant, among other things, that, given the composition, the mechanical properties could be told; that is, that chemical attributes would be convertible into physical.

TABLE 1.—*Extreme Increases in Tensility of Plain Steel Due to Carbon, Heat Treatment, and Cold Working in Pounds per Square Inch*
In Each Case the Steel Has Been Hot-worked

Description	T. S.	Increases Due		
		To carbon	To heat treatment	To cold working
Iron (Swedish).....	45,000			
Iron plus 0.9 per cent. carbon.....	150,000	105,000		
Iron + 0.9 per cent. carbon + heat treatment.....	300,000	105,000	150,000	
Iron + 0.9 per cent. carbon + heat treatment + cold drawing.....	500,000	105,000	150,000	200,000

The tensility of 500,000 lb. per sq. in. is reached in small sizes of wire.

The converse of this is not true, of course, because various combinations of ingredients in different steels may give substantially the same physical or mechanical properties.

The differences existing between the properties of different pieces of steel of *good* quality, and of the same composition, are usually imparted by variations in the working and treatment they have undergone and only in a slight degree by variations in the undetermined impurities or ingredients.

In steels of *inferior* quality, on the other hand, while differences may be due also to working and treatment, they will be caused in a much greater degree by variations in the undetermined constituents.

An individual piece of steel may contain a flaw from which the next piece is free, but the matters that we have to consider are particularly those which relate to the metal itself and affect more or less all parts of the ingot.

So of two steels, perhaps from different sources, one may have better properties with inferior composition while the other has poorer properties with better composition in the common acceptance of the effects of the non-ferrous ingredients. For example, given two steels otherwise similar, the one with higher phosphorus content may be more ductile than the other with lower, just the opposite of what would be expected from the acknowledged effect of phosphorus in steel.

This statement of the inadequacy of the ultimate analysis is not confuted by the practice of using a formula to estimate some of the mechanical properties of some certain grade or kind of steel from its composition at some one steel plant where some particular variation of process is aimed to be followed, and the practice is, within limits, fairly regular; being, relatively speaking, more or less good. In such a case each element may be found to have an influence which is constant enough to be of some value in calculating the properties. But at some other works a different formula is required, and at any works a different

formula is demanded for acid steel from that applicable to basic. It is perhaps the undetermined constituents which chiefly interfere with the general use of some one formula.

Let us state some of the shortcomings of the ultimate analysis:

Shortcomings of the Ultimate Analysis

1. Such an analysis is incomplete. The undetermined constituents may be classified into harmless and harmful; conceding, of course, that they are present only in such proportions as actually exist in commercial plain steels.

The harmless undetermined ingredients include copper, arsenic, nickel, cobalt and aluminum, whose effects on the properties of plain steel are so small as to be of slight importance.

The harmful ingredients are the oxides, silicates and gases. Oxides are probably more prevalent in unfinished, and silicates in finished steel. The gases that injure quality are those which have formed holes in the metal. It seems likely that, in ordinary cases at least, gases in solution have little if any effect on the properties, but evidence on that point is incomplete. We do know, however, that steel may be killed, presumably by keeping its gases in solution, and have good physical and mechanical properties. The lack of a good practical method for determining oxygen in steel is commonly deplored.

2. For the elements which are determined an ultimate analysis is not enough, because, as we will see in Table 3, every one of the elements which enter into the composition of steel occurs at times in more than one state. Thus an element may be present as such, or may be alloyed with iron, or be combined with iron, or combined with some non-ferrous element or elements. Of course, no one element is or can be present in all of the four states. The ultimate analysis does not tell the numerous proximate ingredients formed by the elements in combination. To determine their compositions and amounts would be a substantial step forward, but only a step. In one state an element may have a beneficial and in another a harmful effect on the finished steel.

3. The ultimate analysis tells nothing of any elements, which have been contained or added, which have performed their beneficial service and in so doing have left the metal. A part or all of any silicon, aluminum, titanium, vanadium and manganese which may have been in the molten metal, either residual or added, may have so acted. When alloys have been added for such action the steel may be considered as being alloy-treated but still included in the category of simple or plain steels.

4. The manner in which the ingredients occur and are distributed in the metal is not told and on this its quality largely depends. Full

knowledge of the constitution of the steel must include this, for which end one employs the microscope rather than the beaker. Of internal defects which are not flaws such as welded gas holes, hard spots, collections of sonims and ghosts which more or less impair the mechanical attributes of a test piece in which they or some of them occur, the ultimate analysis gives no indication.

5. The part of the ingot from which the chemist's sample has been taken is often not known, especially when the analysis is made in the endeavor to find out why a certain abnormal steel is as it is, either notably good or notably bad.

A ladle sample properly taken represents the average of the heat well enough but if drillings for analysis are from a finished piece, the part of the ingot from which they came should be known, because of ghosts and regional segregation. If from a part rich in segregate, the figures obtained for carbon, sulfur and phosphorus will be higher than the average, and so mislead.

With enough silicon in the steel to kill it, say 0.25 per cent. or more, or some equally effective gas solvent, the constituent non-ferrous elements segregate but little, and the sample is likely to represent the average fairly well. But in steel partly killed or rimming, segregation is pronounced and the part of the ingot from which the drillings are taken should then be known. The larger the ingot, the more important this is. When less than one foot square, segregation is not as a rule important, and never so when the steel is killed.

6. The ultimate analysis gives no clue to the effects of improper casting temperature, which may be the direct cause of many shortcomings in the steel.

7. If the ultimate analysis were extended to meet all these shortcomings, so that one could know fully and exactly what was in the steel and how, a manifest impossibility, the evaluation or net effect of the whole on the properties might still be beyond us. Any mechanical defect occurring in steel may be due, of course, to causes in no way dependent on or connected with its composition.

Interpretation of Analyses Unsatisfactory

Inability to interpret analyses is an important reason why much that is written about steel is of little value to the steel maker. He reads of results obtained from and of properties possessed by some steel perhaps of a new kind, or for a new purpose, designated by its ultimate analysis, but questions arise in his mind as to what it really is and how it was made. The lack of answers leaves the matter quite ambiguous. He knows that duplicating the analysis will not insure duplicating quality. The analysis alone tells him but little of the details

of the furnace practice by which the steel was produced, and even if the outline of the methods were given trial alone will tell him how to imitate it.

Discrepancies between composition and quality may exist either in sporadic cases, that is in single pieces or heats, or in classes, by which is meant the more or less regular product from particular practices or methods.

At this point let us assert that iron is iron. There is but one chemistry of iron as of every other element. But claims were formerly made that iron in one ore was better than iron in some other. I myself have heard a salesman make such a claim in advocating the merits of certain direct-process wrought-iron blooms made in northern New York to be used as melting stock for acid open-hearth steel. Now, perhaps, such a statement would be too much even for a salesman.

The properties possessed by iron or steel depend on furnace practice, which controls the presence and condition of its non-ferrous ingredients, both known and unknown, on methods of casting, hot and cold working, and heat treatment.

A perfect piece of steel, strictly speaking, never has been and cannot now be made. That is, no piece has desirable properties so supreme that it can be definitely and incontrovertibly maintained that no other piece can ever have better. Considering it only in the annealed state, a perfect piece of steel would contain no harmful chemical ingredient, would be perfectly homogeneous, would corrode uniformly in water and dissolve uniformly in acids, would have no internal stresses, and would be free from flaws or other mechanical defects. As a consequence of all this, it would have the highest mechanical and physical properties its composition could give. If a perfect piece of steel were made it could not be recognized as such and then it would be perfect, that is, best, for only limited and particular purposes.

The number of variables which enter into the production of a piece of steel is so great and the distinguishable degrees of each variable are so numerous, that by permutation the number of cases, each differing noticeably from all the others, is seen to be practically infinite. Then the variations in structural characteristics of each individual piece would multiply the possible number of differing cases by another equally great number. Then, if our imaginary perfect steel were to be heat-treated, a further lot of variables, which are outside our present subject, become active.

As a corollary to this aspect of perfect steel, it follows that no two pieces of steel ever were or can be exactly alike. While all have ills, they differ in any given piece in kind or degree from those in every other piece. Absolute uniformity is unattainable even though practical uniformity may be.

To pursue our subject, let us take first unfinished molten steel in the furnace, following with finished steel, and specifically, unless otherwise noted, basic steel made by the excess-of-pig method, which involves an excess of carbon in the bath metal to be oxidized and boiled out.

UNFINISHED STEEL IN THE FURNACE

The term "finished steel" has different meanings to the different workers who handle it, each considering it finished when he has performed his part. Thus to the steel melter, it is finished when it is solid in the mold, its composition being then fixed. From the roller's standpoint it is finished when in the rolled bar, sheet or plate, the rolling operation changing its form and mechanical properties but not its composition, except that perhaps a little of the carbon has been taken from the skin metal. The forgerman's point of view is similar to the roller's. To the heat-treater, finished steel is that to which more desirable properties are given by heat treatment, together sometimes with incorporation of additional carbon in parts, and is ready for the machine tool or grinder. To the machinist, it is finished only when it is ready to take its place as a part of a machine or structure.

The molten metal of the bath before the final additions contains some carbon, manganese, sulfur and phosphorus as well as a minute quantity of what the chemist reports, if he determines it, as silicon, though, if the charge has boiled, it undoubtedly exists there as oxide of silicon. The other four elements mentioned are those which are or may be determined to enable the heat to be properly finished. In the metal are also certain other oxides, notably those of iron and probably of manganese, which are not determined but are judged indirectly from the character of the boil, nature of the slag, and the fracture of a broken test piece, the content of carbon in the metal being taken into account. In the metal are also certain gases, some in solution and some in process of formation, the latter being the oxides of carbon with possibly some ammonia in the harder steels.

The free boil of the bath is thought to rid the metal to some extent of its non-metallic impurities as well as an important part of the gases, which would be retained if the bath were to lie comparatively dead. The gases then remaining may be easily kept in solution by suitable fresh additions of silicon contained either in crude iron or ferrosilicon.

Additions to the Bath

In finishing a charge of steel, additions of metals, other elements and alloys, are made to it for the double purpose of (1) giving the finished metal the composition and so the properties desired, and (2), preventing or at least diminishing in some degree impending defects, particularly

red shortness, cold shortness and gas holes. These additions to simple steels always contain some carbon and manganese, usually some silicon, and may include aluminum, titanium or vanadium, one or more. Each of these elements, if added, is put in for some definite benefit to the metal. Collectively they are to suppress or at least diminish the harmful effects of oxides, silicates, sulfides, gases and segregation in the finished steel as well as to give the physical and mechanical properties desired. Their individual effects are set forth in Table 2. No two elements have identical roles.

TABLE 2.—*Statement as to Whether or Not Each Element Added to Simple Basic Steel is for Favorable Effect on Each Feature Noted*

Elements	Oxides	Silicates	Sulfides	Gases	Segregation	Mechanical Properties
Carbon.....	No	No	No	No	No	Yes
Manganese.....	Yes	Yes	Yes	No	No	Moderately
Silicon.....	Yes	Yes	No	Yes	Yes	Moderately
Aluminum.....	Yes	No	No	Yes	Yes	No
Titanium.....	Yes	Yes	No	Yes	Yes	No
Vanadium.....	Yes	Yes	No	Yes	No	Moderately

The elements aluminum and titanium, which are not added for direct effect on the mechanical properties, do indirectly have good effect on them by their action on the harmful impurities.

The purposes of the added elements in acid steels are the same except that silicon and titanium are not added to such steel for the elimination of silicates. In acid steel the manganese supplies the basic oxide needed for that purpose.

Carbon added would have a beneficial effect on the oxides in the bath metal by reducing some of them, in part at least, were sufficient time allowed, as it does in certain variations; but as ordinarily added the time thereafter is too short for it to have a noticeable effect on them.

Of the added elements, those which, when combined with oxygen, evolve more heat than iron per unit of weight will be oxidized and so consumed, wholly or in part. Carbon, even though its calorific power of 8100 cal. per kg. when burned to CO_2 is the greatest of all, seems to be an exception to this rule when enough silicon (which has a power of 7595 cal. per kg.) is present to kill the metal, which usually requires 0.25 per cent. or more. The silicon then seizes all the oxygen entering the metal because, in such an environment, and at such a temperature, carbon is oxidized, if at all, only to carbonic oxide (CO) in which case it generates only 2430 cal. per kg.

All the other elements named in Table 2 have greater calorific power than iron and therefore, after being added, are oxidized as stated, so

that an excess of each beyond that desired in the finished steel must be added.

The oxides formed from the additions are precipitated and they then agglomerate or coalesce with each other more or less completely in the well-known manner, the larger ones floating to the surface and merging with the slag, while the smaller, which rise too slowly for this in the time allowed, may be retained in the metal as sonims or non-metallic inclusions. As is the rule in chemical precipitations, agitation or stirring of the bath is helpful to aid the precipitated non-metallic matter to collect into large enough particles to rise; followed by a quiet period to allow them to do so.

Any copper, nickel, arsenic, or other element present or added, whose calorific power when oxidized is less than that of iron, is protected by the latter from oxidation and will persist in the steel. Any sulfur added which contacted with and so combined with iron would also be retained but for another reason; namely, because its affinity for iron at the bath temperature is greater than it is for oxygen. It does, of course, combine with manganese to some extent, the sulfide formed being insoluble in the metal so that its particles either escape to the slag or remain in the metal as sonims.

SOLID FINISHED STEEL

Next we will consider the constitution of solid, finished, though raw steel. Table 3 gives the non-ferrous constituents which may exist in it, showing that a piece even of simple steel may be quite a complex affair. The table contemplates only such concentrations as actually occur. An unduly high percentage of any element may be harmful whatever its state, or at least may render the steel unsuitable for its intended use.

While perhaps any of the gases, solubles and insolubles, shown in Table 3 may exist in any ingot of simple steel, the proximate ingredient which contains each of the five elements of the usual analysis is not ascertained. Table 3 shows therefore yet more of the reasons why such an analysis may give unreliable information about quality. All insolubles in steel, whether gases or solids are harmful. As already stated, the effect of any gases which are in solution in the metal is unknown.

There are also differences in effect of some of the ingredients according to whether they are residues of those in the charge materials, or are added later, as at the end. These effects may be their action on either the gases, or liquids or solids of the molten metal. Thus residual manganese and silicon may have had effect on the non-ferrous solids and liquids in the metal, particularly on the oxides, by furnishing other oxides for fluxing them as already described. The same elements may have checked the boil and so retarded the escape of harmful gases.

The true value of the ultimate analysis must be admitted, of course, for it is used almost universally in assigning steels to uses for which they are suited. But in that case it is supplemented by other knowledge of the metal and particularly of previous results obtained by the method by which the steel was made. No way appears, however, for detecting any abnormal heat of the specified composition which may result from slips or changes in conditions, such as whether the furnace is new or old,

TABLE 3.—Occurrences of Non-ferrous Elements in Simple Ingot Steel
State or Condition and Effect

Elements	Gases		Solubles		Insolubles	
	State	Effect	State	Effect	State	Effect
Carbon.....	CO ₂ , CO in gas holes	Harmful	Carbide CO and CO ₂	Useful ?	Graphite	Harmful
Manganese.....			Alloy	Useful	Oxide Sulfide Silicate	Harmful Harmful Harmful
Silicon.....			Alloy or Silicide	Useful	Silica Silicates	Harmful Harmful
Sulfur.....			Sulfide	Mostly harmful	Manganese Sulfide	Harmful
Phosphorus.....			Phosphide	Mostly harmful	Phosphate	Harmful
Arsenic.....	CO and CO ₂ in gas holes	Harmful	Arsenide	Neutral		
Oxygen.....			Ferrous Oxide FeO	Harmful	SiO ₂ MnO Fe ₂ O ₃ Silicates	Harmful Harmful Harmful Harmful
Hydrogen.....						
Nitrogen.....	Gas holes N, NH ₃ in gas holes	Harmful	Solution Nitride	?	Nitride?	Harmful
Aluminum.....			Alloy	Useful	Oxide Silicate	Harmful Harmful
Titanium.....			Alloy	Useful	Oxide Nitride?	Harmful Harmful

changes in the composition of the crude iron, proportion of crude iron in the charge, changes in the final additions or in fuel, weather or men. The practice at one plant cannot be duplicated *in toto* at some other. Every steel maker in starting to work at some new (to him) plant finds that he has something to learn.

So a formula for calculating properties developed and used at one works cannot safely be assumed to be applicable to steel made at another from different ores, iron and scrap, with other fluxes and additions and under other conditions. It is not so reliable a method as to be a substitute for an actual test.

We will next consider the functions of each of the five elements which comprise the ultimate analysis.

FUNCTION OF CARBON

The great function of carbon in steel is to increase its tensility and hardness, first in its natural state and, secondly, to still further greatly increase those properties in heat-treated steel. This relation of carbon and iron is wholly unique in Nature, no other two elements having anything like the same effect on each other. The usefulness of this effect or relationship to mankind can hardly be overrated.

In annealed hypoeutectic steel, carbon exists combined with iron to form the metal cementite, as is well known. That is its usual state. In the ingot, cementite is likely to be concentrated or segregated in some measure in the interfacial matter between the grains, causing, with other segregated ingredients, the weakness, lack of ductility, and other shortcomings known as ingotism. The resulting structure is characteristic of the higher carbon steels, and the larger the ingot, the more pronounced it is likely to be, because of the slower rate of freezing which affords time for the metal to reject the ferrous compounds and non-metallic ingredients.

For this reason ingots of the order of 50 or 100 tons weight are cast in iron molds to expedite cooling rather than in sand molds which for certain reasons would otherwise be preferred. Proper heating and hot working tend to correct or cure ingotism in ingot steel. In steel castings, suitable annealing will do so by causing the reabsorption of carbon from the cementite by the purer metal of the grains.

In hypereutectic steel, carbon sometimes separates in part from the iron in the form of graphite, in which state it acts like so much foreign matter, highly injurious to the quality. This occurs only in high-carbon steels, perhaps only in those containing over 1 per cent. The higher the carbon, the greater the tendency to separate. Such formation of graphite is generally caused by improper heating; specifically, holding the steel too long a time at a temperature just above the hardening point or about 720° C.

Any carbon combined with oxygen as carbonic acid or carbonic oxide and dissolved in steel of course would be included with the carbonic acid formed by burning the carbon when that element is determined by combustion. The amount, however, is probably too small to be detected by weight. Some investigators of the gases of steel have found so little of these gases that they have concluded that they are insoluble in solid steel. The killing action of silicon and aluminum seems nevertheless to be based on solution of the gases which, if true, indicates that some of them must be in the metal though in what state is a matter for speculation. Evidence about these gases is needed.

Carbon in iron as carbide also has a far greater effect in lowering its fusion point than has any other element—about 90° C. for 1 per cent.

Carbon in steel is also the direct cause of the mushy stage through which steel passes in melting or freezing, as shown by the carbon diagram. With practically no carbon, as in ingot iron, the lines denoting the beginning and end of fusion come together at about 1530° C., at which pure iron melts. They again come together when the carbon content is about 4.3 per cent. and the temperature is 1136° C. Between these limits of temperature and carbon, the metal is a mixture of solid and molten constituents of a mushy or plastic consistency.

FUNCTION OF MANGANESE

Manganese alloys with iron in all proportions, and forms a part of every kind of steel; it has been knowingly used for such purposes for nearly a century. Yet its functions even now are not fully understood. It has some effect in finished steel, but its action or behavior in the metal before and during casting are what largely determine its effect on the product. In studying these effects of manganese one must consider it in the charge materials, as residual, as added, and in the finished steel; then its four known functions in the bath and ingot; then its variations in killed, partly killed and rimming steels; then in the acid and basic processes; then in its minor attributes and in individual steels. Thus by permutation we have a large number of cases of manganese in the different kinds of steel, and in their various stages of progress, at least to review, even though we shall find that many of them are alike. We can refer now only to a few typical or outstanding ones.

In the Charge

If the steel is to be killed, manganese in the original charge materials should average over 1 per cent. For rimming steel it need be but half as much.

General Effects in Molten Steel

Manganese is the great preventer of red shortness, because of its inhibition and elimination of oxides of iron in molten steel, which it accomplishes, first, by intercepting and combining with entering oxygen which might otherwise form them, and second, by reducing those already formed to metallic iron, an equivalent of manganese oxide (MnO) being thereby formed in either case. This oxide is insoluble and therefore harmful to the quality; it calls for elimination, which requires that it be combined with an acid, which, by fluxing it, will form a product that will collect into particles large enough to float out. In the acid process there is usually emulsified silicic acid in the bath metal, which serves this purpose. In the basic process silicon should be added, which will, by its oxidation, form the acid needed.

In molten steel manganese has some solvent power on the contained gases or some of them, though not great enough to be of much help in preventing the formation of gas holes and so killing the metal. Through this property, together with its superior affinity for oxygen, it has some quieting effect on the boil during decarburization, exercising therein a somewhat obscure function. It also lowers slightly the fusion point of the metal.

Manganese may also effect a partial elimination of sulfur from the molten metal by decomposing sulfide of iron, forming insoluble sulfide of manganese, a part of which may reach and mingle with the slag; with which, however, it is immiscible.

Residual Manganese

With the percentages mentioned of manganese in the charge, some will remain in the metal at the end of the heat, which shows that some has been present throughout the melt where it has served to limit the concentration of iron oxides in the metal. For the best results, this continued presence of manganese is essential whatever the kind of steel.

For killed steel there seems to be no reasonable upper limit to the amount of residual manganese that may be in the metal at the end without harm, if not with benefit to the quality. I say "there seems to be," because, as manganese has a quieting effect on the boil and a proper boil may aid in the elimination of harmful non-metallics and gases, it is at least conceivable that the quiet boil resulting from excessively high residual manganese, say over 0.5 per cent., may be prejudicial to the quality of the product because of these retained impurities. This is, however, conjectural. Every steel melter wants what he considers a nice boil, or sequence of boils, of the bath, but can give no reason for his desire except that he finds that with such boils his results are best.

For effervescing rimming steel less residual manganese is needed than for killed and therefore less in the charge materials. The best percentage is not yet definitely determined, but in the common grades of basic steel lies probably between 0.10 and 0.20 per cent. It should be distinctly less than for killed steel as a more lively boil is needed. To decide what is best we need among other things full knowledge of the gases of low-carbon and low-silicon unfinished steels; both what they are and how they behave. This knowledge is almost wholly lacking. Acid steel that rimmed in well has been made with but 0.01 per cent. residual manganese.

It seems probable that a proper boil eliminates some of the gas which forms skinholes in the rimming steel ingot. On the degree of concentration of this gas in the metal depends the depth in from the surface at which it begins to separate in bubbles, which form holes, and on that depth much depends.

With high residual manganese, say 0.25 per cent., carbon in the metal may be oxidized only slowly and with difficulty below 0.12 per cent. This seems to be due to the power of manganese to seize much of the oxygen entering the metal, even though its calorific power when oxidized to MnO (1653 cal. per kg.) is less than that of carbon when oxidized to CO (2430 cal. per kg.).

In basic steel residual manganese is likely to be reinforced by some reduced from oxide in the slag by carbon in the metal. Some of this oxide may have come from ore added to supply oxygen. As much as 0.2 or 0.3 per cent. of manganese may be so reduced. In acid steel this is much less likely to happen, and when it does it is in smaller amount, less than 0.10 per cent. and only in charges containing at least 0.5 per cent. carbon, which are held molten in the furnace for hours without addition of oxides.

Added Manganese

Manganese added at the end of a heat may not always clean the steel of oxides as completely as it should, either because the amount is inadequate, which is seldom the case, or the time elapsed after its addition is too short. Of the manganese added, some is oxidized by oxide of iron in the slag and is so wasted except for its equivalent of iron, which it reduces to the metallic state. Some is oxidized by oxides of iron contained in the metal in exercising its therapeutic function and that part is of the utmost utility and importance. The remainder, which comprises the bulk of it, is alloyed with the iron of the charge to give the concentration desired in the finished steel. This excess, which is necessary to insure that the destruction and elision of the oxides of iron in the metal be reasonably prompt and complete, together with the residual manganese and any contained in silicate and sulfide sonims in the sample, the chemist reports as manganese. The analysis tells the amount but not its story.

Added Manganese in Rimming Steel

To low-carbon effervescing steel to be rimmed manganese is added at the end solely as a physic, or specifically to prevent red shortness, and not for any effect on the mechanical properties. For most uses the proportion aimed at should be about 0.4 per cent. in the finished steel. More than that, say over 0.5 per cent., may, through its solvent power on the gases, or some of them, check their escape and so inhibit the churning action in the mold which is necessary for it to rim in properly. As a consequence, the steel may rise in the mold, which is always a sign that gas holes are forming. If it rises immediately after teeming these are harmful skinholes. In the pig and scrap method, when ore has not been used for decarburization, less manganese will serve, or from 0.25 to 0.30 per cent.

If the boil in the furnace at the end is unduly vigorous, a small increase of manganese added may be beneficial, while if the boil is too gentle, ore is needed.

Steel for making welded pipe must rim in acceptably, though as regards welding a considerable latitude in manganese, say from 0.35 to 0.45 per cent., is allowable.

Low-carbon steel cast too hot may rise in the mold and be worthless because of skinholes, even though the manganese be as desired.

Finished Manganese

In well made finished simple steel of any grade, manganese raises tensility about 150 lb. per sq. in. for each point contained. Increasing manganese, within limits, enables steel to endure without harm a higher temperature when heated in a furnace for hot working. This latter effect, however, may be indirect and due to better deoxidation by the higher manganese, and consequently more complete freedom from red shortness, rather than to the manganese itself.

In killed and partly killed finished steel which is not to be hardened and tempered, the manganese content, which is usually the greater part of 1 per cent., is more than has really been needed for good quality, the excess being added to make up in a way for insufficient time after the final additions were made.

For steel to be hardened and tempered as it is in many tools, manganese should be lower; the percentage depending on the bulk of the piece to be quenched and the carbon content. When the steel has over 0.9 per cent. of carbon and the piece is massive, say several inches thick, the manganese should not exceed 0.3 per cent. Such steel must be made with due regard to quality and treatment and ample time must be allowed for the manganese to do its work. With higher manganese the piece is in danger of cracking in the hardening quenching. This danger becomes less as the carbon and thickness (or either of them) are diminished.

Lower manganese than expected in the finished steel, if due to an overoxidized bath, may be a sign of a tendency toward red shortness, while if due to increased lapse of time after the final additions were made it may indicate well deoxidized steel. In the latter case it may be due to an intelligent and successful effort to compensate for an overoxidized bath by allowing additional time for the manganese to do its work, more, but perhaps not enough, being added usually in an effort to compensate for that oxidized in the metal and slag. So a steel abnormally low in manganese may be of ordinary, or inferior, or excellent quality. The percentage of manganese determined by the chemist does not tell which.

In the olden days much poor steel was made in the effort to give it, when finished, the specified content of manganese, and, at the same

time, to be economical of ferromanganese. So the time after the manganese was added was likely to be cut too short for its proper effect, and a red-short tendency resulted. It was not generally recognized that it was the manganese utilized as noted which cured red shortness rather than that remaining in the finished steel.

The practice of adding manganese in the ladle instead of in the furnace, as had been customary, was introduced at that time for purposes of economy. When manganese is so added it must be done intelligently, with due regard to its functions, or the quality of the product may be inferior.

Four Functions of Manganese

The four principal known functions of manganese in the acid process are then:

1. To cure its red-short tendency by decomposing oxides of iron in the metal.
2. To provide a basic oxide for fluxing emulsified silicic acid in the molten metal.
3. To eliminate some of the sulfur.
4. To give the specified composition in the finished steel.

The solvent effect of manganese on the gases, particularly those of rimming steel, seems to be that in the bath it checks the boil, while in the mold it retards both effervescence and the separation of skinhole gas.

In the basic process the basic oxide of manganese formed needs the presence and action of an acid to eliminate it.

FUNCTION OF SILICON

The actual state of silicon in steel is more uncertain than that of any of the other four elements, particularly when the amount is small or under 0.05 per cent. Table 3 shows that it may be there as alloy, or silicide, or oxide, or as a silicate, one or more, but the analysis does not tell which. Any present may be residual from that in the charge materials, that is, if the charge has not boiled, or may have been added at or near the end. As already noted, some may have been added which has done its work improving quality, and after so doing has left the metal, thereby escaping the chemist.

Silicon is added to simple steel as a curative, and in so serving it may play one or more of several parts which we will note in detail later. It has also been thought to increase the resistance to corrosion of plain steel as it does in crude iron when high—up to 10 per cent. or more. If it does so act in steel it is a matter of minor importance, since the superior abilities of copper and chromium to that end have become recognized.

The actions and so the effects of silicon are so different in acid and basic steels that each must properly be considered by itself.

Residual Silicon in Acid Steel

Silicon in acid steel may be residual, or added, one or both. Residual may be present in such steel in either of two forms, one of which occurs only with low carbon and the other only with high.

When there is more than 0.01 per cent. silicon in low-carbon acid steel to which no silicon has been added, and the bath in the furnace has boiled, as it practically always does when the carbon is low, that silicon will pretty surely be in the form of silica remaining from that formed by oxidation of silicon in the charge materials. Most of the silica so formed has, of course, left the metal and entered the slag, but some has persisted in the metal because it was infusible at the bath temperature and there was not enough oxide of manganese present to flux it. This happens when the original charge has too little manganese, say less than 0.3 per cent., and no metallic manganese has been added. All of that element may have been oxidized and eliminated before all the silicon has been, for in such a case there will be sufficient emulsified silica in the metal to flux any oxide of manganese therein. Then any silicon oxidized later remains for the most part in the metal until the final addition of manganese is made. A part of that is oxidized and the oxide fluxes a part of the remaining silica, but 100 per cent. cannot be so eliminated. That remaining is highly injurious to quality, tending to make the steel red short when hot worked, and lessening ductility.

In making acid high-carbon steel such as that for springs, with from 0.8 to 1.0 per cent. of carbon, by the pig-and-scrap method, there must be a relatively large proportion of pig in the charge to provide the carbon desired. In that method no excess of carbon is intentionally added, but the quantity of pig to give that desired may also contain so much silicon that when all is melted the metal will still hold 0.3 or 0.4 per cent., which is enough to make it lie dead in the furnace. If the carbon content is right and the silicon not objectionable, the charge may be cast with that amount of residual silicon.

Some makers of springs object, however, to so much silicon in their steel, as it does not suit their tempering methods, and sometimes the carbon is much too high for the specification. In either case the silicon in the metal must be nearly all oxidized by added ore. Carbon cannot be oxidized in the presence of much silicon. If it is but little too high, say less than 0.1 per cent., it may be diluted by the addition of low-carbon scrap steel or wrought iron and made right without ore additions. When residual silicon in such a heat is unduly high, carbon is likely to be high also, as the case usually results from exceptionally quick melting and consequently less waste of the charge materials by oxidation.

Most unusual, but yet possible, is the case of an acid steel with high residual silicon, say from 0.6 to 1.0 per cent. or even more, and the ingot

badly infested with gas holes. Such a heat is made only in the pneumatic converter, when the charge in blowing becomes excessively hot, due usually to too much silicon in the crude iron. At an unduly high temperature, apparently around 1650° C., the affinity of silicon for oxygen is much weakened, if not destroyed, while that of carbon for oxygen is increased. Consequently, in a heat which reaches the critical temperature, the carbon in the iron is all burned out while a considerable part of the silicon remains in the metal unoxidized when the flame drops, which it does when the carbon is all consumed. The major part of the silicon is indeed oxidized before the temperature reaches the critical point but the unburnt residue may be as stated. The copious volumes of carbonic oxide and other gases formed or otherwise present have saturated even that great amount of silicon with all it can hold, so that as the metal solidifies in the mold enough is set free to form a profusion of gas holes as described. This case never occurs in open-hearth steel.

Added Silicon in Acid Steel

Silicon is added to plain acid steel (1) to kill it more or less perfectly and (2) sometimes as a step toward deoxidation. For killing, the proportion is usually something between 0.2 and 0.6 per cent. With less than 0.2 the steel is likely to be but partly killed unless the silicon is reinforced by the addition of some other gas solvent such as aluminum or titanium. With low carbon rather more solidifier is needed than with higher carbon.

This solidifying effect is sometimes called deoxidizing, but incorrectly so, as acid steel killed with silicon alone may be far from being deoxidized; being too red short to be of any use. Partial deoxidation with silicon is the reaction between that element and the oxides of iron in the unfinished steel resulting in the formation of silicic acid and the liberation of metallic iron. It takes place quickly after the silicon is melted and incorporated with the metal. The silicic acid formed is an infusible, non-metallic impurity which if permitted to remain in the metal is, like residual silica, more harmful to the properties than the oxides of iron it has decomposed. To eliminate it a basic oxide, or, particularly, oxide of manganese is needed to combine with and flux it, so that its particles may coalesce and escape.

For killing steel no time is required after the silicon is melted and mixed with the metal, as its solvent power becomes immediately effective. Probably no chemical action is involved. For this reason a large part may properly be added in the ladle. Some of the silicon added for killing the steel will, however, be oxidized, even with the addition of plenty of manganese to do the deoxidizing, and form silicates which, being minutely comminuted, leave the metal but slowly. For this reason a part of the

silicon should be added in the furnace so that the silicates so formed may have ample time to escape as completely as the desired quality of the steel demands. The power which silicon has to prevent the formation of gas holes and so to kill steel is explained in two ways:

1. If the gas to be suppressed is a compound, particularly carbonic oxide, which is the most plentiful gas in ordinary molten steel, it may conceivably be decomposed by the silicon with the formation of silicic acid, which is insoluble in the metal, and carbon which is soluble therein and therefore absorbed. Neither of these products is gaseous at the temperatures of molten steel. The action of silicon in preventing the formation of carbonic oxide is of course equivalent to its decomposition.

2. According to the other explanation, the silicon keeps the gaseous compounds in solution and so prevents their separation on freezing and thus their formation of gas holes. This usually, if not always, seems to be the correct explanation. Any elementary gas, such as hydrogen, must of course be kept in solution by the silicon.

Sometimes a part of the silicon in acid steel has been reduced from silica in the bottom or slag by the action of carbon in the metal. This may be as much as 0.1 or 0.2 per cent., or nearly enough to kill the steel. It happens only when the carbon content is comparatively high, or over 0.5 per cent., and the bath is held for some hours in the furnace without the addition of oxide of iron in any form. The slag then becomes strongly acid and vitreous, containing from 55 to 60 per cent. of silica. The bath is almost dead. Bubbles are few and each one bursts with an audible snap through the tough slag. Silicon in steel so reduced is a guarantee that the metal has been well purified of its oxides. In the acid electric furnace, under similar favoring conditions a still more rapid and more prolific reduction of silicon is likely to take place.

In the old method of melting crucible steel, which is still practiced to some extent, this manner of reducing silicon from the siliceous environment was the only way in which the metal acquired that element and was killed. Up to 0.6 per cent. or even more was sometimes so reduced. The average was perhaps between 0.3 and 0.4 per cent., which was quite enough to kill the steel, the maximum mentioned above being more than was needed.

Assuming apparently that silicon found by the chemist meant always the one thing, some crucible-steel makers have hastened the process and shortened the time for making a melt by adding the desired percentage of silicon to the charge in the form of ferrosilicon. They either put it in the pot with the cold charge or add it after the charge is melted. Whether these accelerated methods are equally good or whether either is as good as the old way, those who practice them may or may not be able to demonstrate. The silicon is undoubtedly alloyed with the iron or combined as silicide in all three cases alike; but the time required for

killing in the old way, which is largely saved by the newer methods, and the boiling action which the metal has when first melted, if the silicon be low, may be needed for, and aid in, cleaning the metal of its non-metallics and also possibly of some of its deleterious gases. This time and boil may tend to impart superior quality to the steel.

Holding the molten charge for killing by the absorption of silicon from the slag and walls is more practicable with clay pots than with those made of graphite because, during that period, too much carbon would be absorbed from the latter. This would waste the crucible, unduly shortening its life; and, as the amount of carbon thus entering the steel would be variable, the product might be too irregular in its content of that element for commercial practice.

In well made, finished simple steel, silicon raises tensility about 150 lb. per sq. in. for each point contained. This effect is sometimes not attained in steels of poorer quality, being masked by more or less abundant non-ferrous or non-metallic impurities remaining in the steel. Silicon is not added primarily to give increased strength to plain steel, carbon being six or seven times as effective for that purpose, which indicates that silicon is alloyed with the iron and not combined as carbon is.

Residual Silicon in Basic Steel

In basic open-hearth steel neither of the cases of residual silicon which occur in acid steel as noted will be encountered. Any notable amount of silicon found in basic steel has been added to it at or near the end. As residual it is rarely if ever present; certainly never in any steel from which an excess of carbon has been eliminated by oxidation, as is almost universally the case; for, as we have noted in the case of acid steel, practically all the silicon must be oxidized before much carbon can be.

There is not enough acid to flux all the basic oxides of iron and manganese which impregnate the metal at the end. The basic environment, both hearth and slag, absorbs substantially all the silica from the metal early in the course of the heat. An acid is therefore called for, as we have noted.

The chemist may indeed report 0.005 per cent. in the finished steel, an exceedingly small amount, which, if really present, may be in the form of emulsified silica or silicate. He cannot tell if the silicon he finds is combined with iron or oxygen.

Added Silicon in Basic Steel

Silicon may or should be added to basic steel shortly before the end for several different purposes, as follows:

1. To prevent further oxidation of carbon. This addition is made in the furnace and is to hold the charge as it is chemically, excepting the oxidation of the silicon itself, while the bath metal is being analyzed. Oxygen which reaches the metal after the added silicon is incorporated with it is seized by that element, none being available for oxidation of carbon.

2. To help deoxidize the unfinished steel. For this purpose, which is to minimize red shortness, any quantity up to 0.15 per cent. may be added in the furnace. While manganese is the great deoxidizer, silicon is generally held to aid. Alone, however, unless considerable residual manganese is present, it is not efficacious.

3. To form an acid which will flux and so promote the elimination of basic oxides from the molten metal. This important function must be considered together with that of deoxidizing which it follows. The oxides of manganese and iron present are both basic and have no affinity for each other. They need an acid to flux them and so enable them to coalesce and escape. Any silicon added up to 0.15 per cent. to stop the boil of a basic bath is largely oxidized during the time the charge is held, even though the boil may not have recommenced; 0.02 per cent. of silicon so remaining is enough to keep the bath quiet even though 0.12 or 0.15 per cent. was required to practically stop the boil. The silicic acid formed serves to elide the basic oxides as noted. Ordinary ferromanganese or spiegel usually contains some silicon which is oxidized before the manganese and serves the same way after addition. It is usually better, however, to insure an ample supply by adding additional silicon; which, however, in freeing the metal of oxide may have left it, so that it will not be found by the chemist.

In acid steel, if additional silicon is required, the need is not so pressing or universal as in the case of basic.

4. To prevent the formation of gas holes. To avoid undue loss, and so to do this effectively, the silicon for this must be added in the ladle. It acts much as it does in acid steel but more is oxidized in the metal and slag. The oxide of iron in the latter tends to oxidize carbon as well, making more gas for the silicon to hold in solution; so a somewhat larger percentage of it is needed to kill basic steel than acid steel, or about a fifth more.

The first function is a manufacturing convenience. The other three are all curative. In a moderate degree, silicon also lowers the fusion temperature and gives heat to the metal by its oxidation.

Time Required for Different Functions of Silicon

After an addition of silicon is melted and mixed with the molten metal, the time required for it to exercise its various functions varies greatly:

1. To retain gases in solution and to check further oxidation of carbon the action is immediate—practically instantaneous.

2. To decompose oxide of iron in solution is a matter of seconds.

3. To form silicates of iron and manganese is a matter of minutes. This includes the fluxing and merging of emulsified oxides by the fluid silicates first formed.

4. To clean the metal of silicates completely is a matter of hours.

Some effects of high temperature on the affinities of silicon, carbon and oxygen for each other in iron metallurgy are indicated in Table 4. The reaction is reversible according to the formula

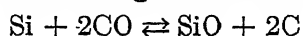


TABLE 4.—*Chemical Relations of Silicon, Carbon and Oxygen as Affected by High Temperatures*

TEMPERATURE, DEGREES CENTIGRADE	BEHAVIOR
700	Silicon reduces CO in the laboratory, forming SiO ₂ and liberating carbon.
1550	Silicon prevents the formation of CO in the open-hearth bath and so tends to kill the metal.
1600	Carbon reduces silicon moderately from slag and hearth in the acid, open-hearth crucible and electric processes.
1650	Carbon prevents oxidation of silicon in the bessemer blow. Perhaps reduces it.
1900	Carbon reduces silicon freely from its oxide in the blast furnace.

Below the equilibrium temperature of the reaction of silicon with carbonic oxide, which seems to be in the neighborhood of 1600° C., the affinity of silicon for oxygen is stronger than that of carbon, while above it the affinity of carbon excels.

It appears, therefore, that the percentage of silicon reported by the chemist requires supplementary information to give its full meaning. To appreciate its significance one must know how it came to be in the steel.

FUNCTIONS OF SULFUR AND PHOSPHORUS

The percentages of sulfur and phosphorus usually stand for what they are. The only points in doubt are whether they are residual or added. If the latter, they are thought to work less damage to the ductility of the steel than if residual.

Added phosphorus is held to have some deoxidizing effect on the steel and, if there is a deficiency of silica, its oxide may serve as an acid to flux basic oxides in the metal, as does silica. This action may be the basis of the commonly held opinion that added phosphorus does not make the steel as cold short as residual; but as ferrophosphorus always contains some silicon, it may be that phosphorus gets credit that rightfully belongs to silicon.

When more than 0.05 per cent. of either of these elements is present in basic steel, some has probably been added to promote machinability, or, in the case of phosphorus in thin sheets, to keep them from sticking together too strongly when rolled doubled. An excess of each is needed in either case because a part of the phosphorus so added to basic steel is oxidized, while a part of any added sulfur is volatilized.

SUBJECTS FOR RESEARCH

During this talk we have considered some things the ultimate analysis does not tell us regarding the make-up of steel, including:

1. The genesis, composition, behavior and effects of the various gases, both in gas holes and in solution.
2. The genesis, composition, manner of occurrence and effects of the various sonims.

These are subjects worthy of research. But let us urge that in studying them the history of the steels dealt with, particularly the furnace practice by which they were made, in all details, be noted and recorded as fully and as clearly as can be done by language, to the end that the man in the shop may be enabled to utilize the results in his work—to apply the good and avoid the bad.

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A. I. M. E. Technical Publications, 1928

[Separates of all the *Technical Publications* published in 1928 are available at Institute headquarters. All the papers are on file in public, university and technical libraries, and when so indicated in the list, may be found either in the TRANSACTIONS or in the special volumes issued by the Institute.]

METAL MINING

TECH- NICAL PUBLI- CATION No.	TITLE	AUTHOR	MEETING	TRANS. VOLUME AND PAGE*
50	The Eötvös Torsion Balance Method of Mapping Geologic Structure	D. C. Barton	New York Feb., 1928	
52	A Cartographic Correction for the Eötvös Torsion Balance	C. A. Heiland	New York Feb., 1928	
61	Liquid-oxygen Blasting at Chuqui- camata, Chile	H. C. Schultz, F. K. Middleton Hunter	New York Feb., 1928	76, 108, 123
64	Working Methods of Practical Geo- physics	Hans Haalck	New York Feb., 1928	
65	Experiments with Eötvös Torsion Balance in the Tri-State Zinc and Lead District	P. W. George	New York Feb., 1928	
74	Gravity Surveying in Great Britain	H. Shaw	New York Feb., 1928	
75	The Computation of Eötvös Gravity Effects	E. Lancaster-Jones	New York Feb., 1928	
82	Earth-resistivity Measurements in the Lake Superior Copper Country	W. O. Hotchkiss, W. J. Rooney, James Fisher	New York Feb., 1928	
83	Mining Methods and Records at the United Eastern Mine	Roy W. Moore	New York Feb., 1928	76, 56, 90
98	Recent Results in Electrical Pros- pecting for Ore	Hans Lundberg	New York Feb., 1928	
101	Relation between the Oxygen Bal- ance and Propulsive Strength of Eight Gelatin Dynamites	G. St. J. Perrott, N. A. Tolch, J. E. Crawshaw	New York Feb., 1928	
102	Effect of Oxygen Balance of Gelatin Dynamites on the Gaseous Prod- ucts of Detonation	G. W. Jones, G. St. J. Perrott	New York Feb., 1928	
108	The Financial Report of a Min- ing Company—Its Content and Meaning	Henry B. Fernald	New York Feb., 1928	
116	What Duty to Support the Surface Does a Subsurface Owner Owe?	R. G. Bosworth	New York Feb., 1927	
136	Comparison of Branch Raise and Combined Shrinkage and Caving Methods	C. A. Mitke	Los Angeles Sept., 1928	76, 93, 103
141	A Mining Company Balance Sheet	G. F. Wolf	New York Feb., 1928	

* Volume numbers are bold face; page numbers of papers, Roman; and page numbers of discussions, italics. Where no number is given, paper has not appeared in the TRANSACTIONS or a special volume. Discussions are printed following the respective papers.

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155	Mine Development and Under- ground Construction of Andes Copper Mining Co. at Potrerillos, Chile	I. L. Greninger	New York Feb., 1928	
MILLING AND CONCENTRATION				
41	Gas Sorption in Flotation	A. S. Adams	New York Feb., 1928	B28 ,* 216, 225
46	Crushing and Grinding, I.—Surface Measurement of Quartz Particles	John Gross, S. R. Zimmerley	New York Feb., 1928	
126	Crushing and Grinding, II.—The Relation of Measured Surface of Crushed Quartz to Sieve Sizes	John Gross, S. R. Zimmerley	New York Feb., 1929	
127	Crushing and Grinding, III.—Rela- tion of Work Input to Surface Produced in Crushing Quartz	John Gross, S. R. Zimmerley	New York Feb., 1929	
132	High-efficiency Motor Generator Sets for Electrolyzing Leaching- plant Solution	David Hall	Los Angeles Sept., 1928	
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47	The Manufacture of Some Foreign Rails	C. W. Gennet, Jr.	New York Feb., 1928	C28 ,† 177, 102
48	Effect of Annealing on Cold-worked Single Crystals of Silicon-ferrite	Hugh O'Neill	New York Feb., 1928	C28 , 229, 248
59	Application of a High-vacuum In- duction Furnace to the Study of Gases in Metals	P. H. Bracc, N. A. Ziegler	New York Feb., 1928	E28 ,‡ 544, 564
62	Utilization Problems of Metallurgi- cal Limestone and Dolomite	Oliver Bowles	New York Feb., 1928	C28 , 87, 100
73	Improvements in Blast Furnace Construction	J. P. Dovel	New York Feb., 1928	C28 , 30, 33
88	Effect of Temperature on the Solu- bility of Iron Oxide in Iron	C. H. Herty, Jr., J. M. Gaines, Jr.	New York Feb., 1928	C28 , 142, 154
94	Steel for Aircraft Construction	E. A. Richardson	New York Feb., 1928	C28 , 157, 175
96	The Constitution of the Iron-silicon Alloys	M. G. Corson	New York Feb., 1928	C28 , 249, 296
100	Critical Points in Chromium-iron Alloys	A. B. Kinzel	New York Feb., 1928	C28 , 301, 307
106	A One-ton Acid Open Hearth and Some Experimental Results	C. E. Meissner	New York Feb., 1928	C28 , 103, 111
109	Operation of Blast-furnace Plant of Columbia Steel Corp'n. at Iron- ton, Utah	W. R. Phibbs	New York Feb., 1928	C28 , 62, 65
111	Rate of Carbon Elimination and Degree of Oxidation of the Metal Bath in Basic Open-hearth Prac- tice	Alexander L. Feild	New York Feb., 1928	C28 , 114, 130
112	Production of High-alumina Slags in the Blast Furnace	T. L. Joseph, S. P. Kinney, C. E. Wood	New York Feb., 1928	C28 , 66, 85
125	Cleaning Blast-furnace Gas	Arthur J. Boynton	New York Feb., 1928	C28 , 36, 61
139	The Neumann Bands in Ferrite	C. H. Mathewson, G. H. Edmunds	Philadelphia Oct., 1928	C28 , 311, 333

* **B28** indicates inclusion in FLOTATION PRACTICE.† **C28** indicates inclusion in IRON AND STEEL TECHNOLOGY IN 1928.‡ **E28** indicates the *Proceedings* of the Institute of Metals Division, 1928.

NON-FERROUS METALLURGY

TECH- NICAL PUBLI- CATION No.	TITLE	AUTHOR	MEETING	TRANS. VOLUME AND PAGE
49	Suggested Improvements for Smelting Copper in the Reverberatory Furnace	G. L. Oldright, F. W. Schroeder	New York Feb., 1928	76, 442, 461
68	The Contamination of Metal Scrap, Its Effect on the Value, and Suggested Means of Control	Carl O. Thieme	New York Feb., 1928	E28, 711, 717
69	Thé Waelz Process (With Discussion)	R. Hoffmann	New York Feb., 1928	76, 537, 548
72	Reduction of Roasted Cassiterite Concentrates	W. W. Loo, E. F. Kern	New York Feb., 1928	76, 494, 522
81	Sampling and Evaluating Secondary Non-ferrous Metals	T. A. Wright	New York Feb., 1928	E28, 672, 688 76, 644, 660
84	Refining and Melting Some Platinum Metals	J. O. Whiteley, C. Dietz	New York Feb., 1928	76, 635, 648
87	Purification of the Six Platinum Metals	Edward Wichers, Raleigh Gilchrist, William H. Swanger	New York Feb., 1928	76, 602, 631
92	Modern Non-ferrous Secondary Metal Producer	Don C. Blackmar	New York Feb., 1928	E28, 700, 709
95	Use of the Noble Metals for Electrical Contacts	E. F. Kingsbury	New York Feb., 1928	E28, 804, 822
97	Non-ferrous Secondary Metals Recovered in the United States	J. P. Dunlop	New York Feb., 1928	E28, 660, 669
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Magnetometric Investigation of Gold Placer Deposits near Golden, Colo.	C. A. Heiland, Wm. H. Courtier	Boston Aug., 1928
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